MINING ENGINEERING

FEBRUARY 1950

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WILFLEY pumps are trouble-free. They operate without attention...carry the load easily and economically. Production records in today's most modern chemical plants prove the cost-saving dependability of these highly efficient pumps. 10- to 2,000-G.P.M. capacities; 15- to 150-ft. heads and higher. Individual engineering on every application. Write or wire for specific details.

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WILFLEY *Acid* PUMPS

Incorporating
MINING and METALLURGY
MINING TECHNOLOGY
COAL TECHNOLOGY

MINING ENGINEERING

VOL. 187 NO. 2 FEBRUARY 1950

In This Issue

EDITORIAL

175 Something Left to be Done

ARTICLES

178 Cerro Bolivar—Saga of an Iron Ore Crisis Averted
193 Where Does Coal Go From Here?
199 The Engineering Curriculum

FEATURES

173 Reporter
176 It's Everyone's Business
198 Book Reviews
165 Manufacturers News
200 New Bureau of Mines Personnel
170 Authors in This Issue
169 Letters to the Editor
160 Personnel

TECHNICAL ARTICLES

201 Problems of a Mine Mechanization Program
204 Health and Safety Practices at Pioche
207 Faster Calculation of Plane Triangulation Systems
217 Developments in the Application of Activated Carbon to Cyanidation
223 Continuous Countercurrent Decantation Calculations
226 Effect of Conditioning on Flotation of Chalcotite
239 Canadian Deposits of Uranium and Thorium
256 Quantitative Efficiency of Separation of Coal Cleaning Equipment
265 Some Factors Influencing Performance of Single Retort Underfeed Stokers
275 Research in Coal Geology
279 Ground Water in California
285 Effect of Freight Rates on Marketing Northwest Industrial Minerals
289 Fuel Economy in the Lepol Kilns
292 Industrial Mineral Economics and the Raw Materials Survey

NEWS

297 AIME Annual Meeting News
299 Executive and Finance Committee Meets
299 Coming Events
300 The Drift of Things
302 Student News—Prize winners
303 Personals
304 Local Section News
304 Obituaries
307 Membership
312 Advertisers' Index

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***The term "Type R" by which these Allis-Chalmers crushers have been known has been changed to "*Hydrocone*." "Hydro" denotes the use of a static liquid, such as oil, used in the *Hydrocone* crusher for supporting and adjusting the height of the crushing cone. The principle of operation has not been changed.**

TO MEET new requirements for fine crushing in a larger range of capacities, Allis-Chalmers offers the expanded line of *Hydrocone* crushers, in sizes up to 1784 (17-in. receiving opening; 84-in. diameter cone) with coarse, intermediate or fine crushing chambers.

In the *Hydrocone* design the crusher's main shaft is raised or lowered hydraulically (push-button operation on the larger sizes) permitting quick, accurate product size adjustment without stopping the crusher. An Automatic Reset device lowers the entire crushing head hydraulically to pass uncrushable materials.

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- Simplicity of design reduces maintenance costs substantially.

Get the complete facts about *Hydrocone* crushers — in sizes 122 to 1784 — from the Allis-Chalmers representative in your area. A-2933

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Texrope Drives

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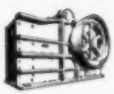

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Kilns, Coolers, Dryers

Gyratory Crushers

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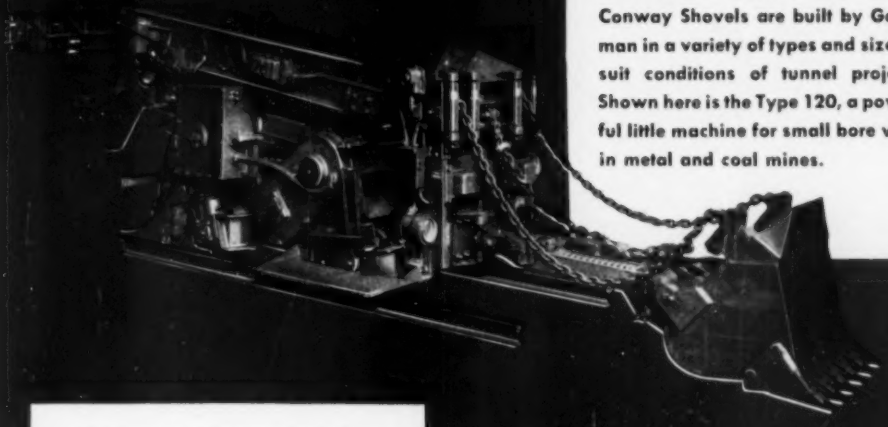
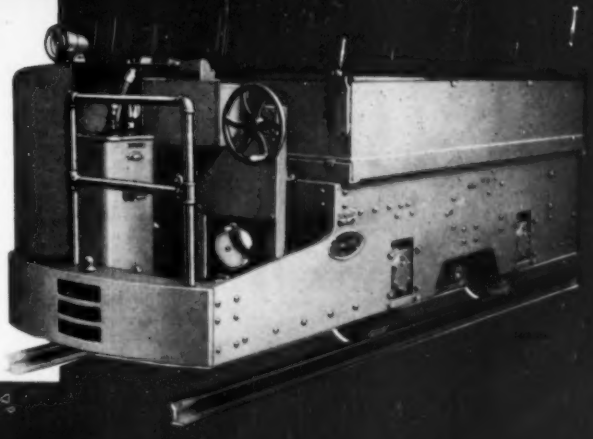
CONWAY MUCKING SHOVELS

Conway Shovels are built by Goodman in a variety of types and sizes to suit conditions of tunnel projects. Shown here is the Type 120, a powerful little machine for small bore work in metal and coal mines.



GOODMAN LOCOMOTIVES

Fast hauling of heavy loads is provided for Conways by Goodman locomotives. They are available in trolley, storage battery or combination types. The storage battery unit illustrated here is in the 10 ton class and is powered by two 45 hp motors for operation at 110 volts.



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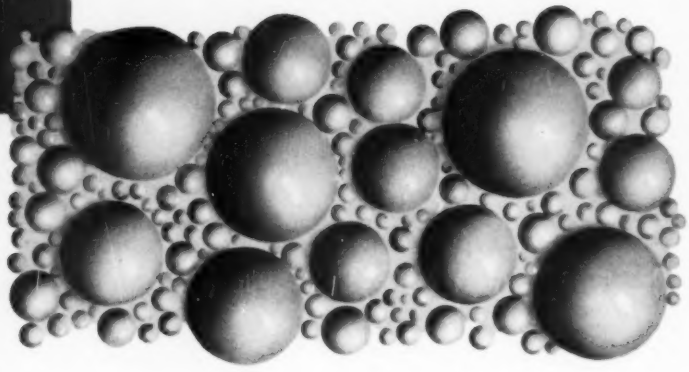
"Over Twenty-Five Years' Experience in Producing Xanthates for Metallurgical Use."



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- **Z-3—Potassium Ethyl Xanthate**
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*From Sharples Amyl Alcohols

Time and experience have demonstrated that for optimum results in the flotation treatment of substantially all sulphide ores, as well as some oxidized ores and ores containing native metals, Bear Brand Xanthates are the cheapest and most efficient collectors now available.

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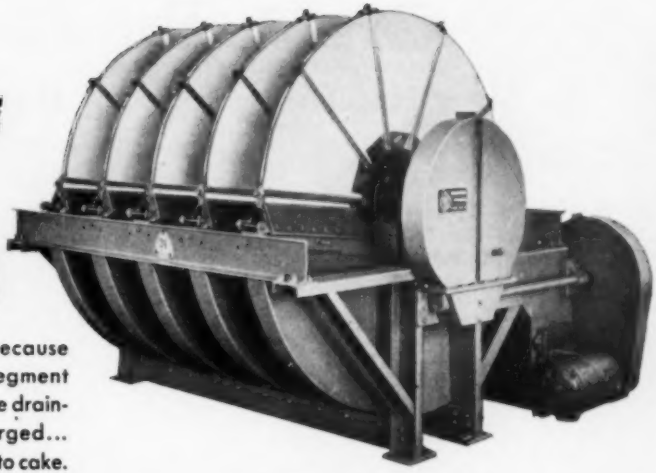
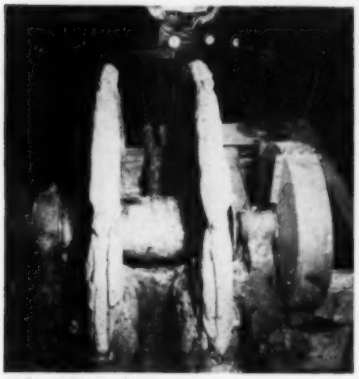
DRIER FILTER CAKE

with

DENVER DISC FILTER



Thicker and ***Drier*** filter cakes because PATENTED grooves in filter segment provide ***complete, positive*** filtrate drainage just ***before*** cake is discharged... prevents "blow-back" of filtrate into cake.



This photo shows:

Zinc feed	(45% moisture)
Filter cake	(8% moisture)
Feed per hour	1¼ Tons
Filter size	4'-Two Disc

FREE FILTERING TESTS

The Denver Ore Testing Division will conduct tests without charge to determine:

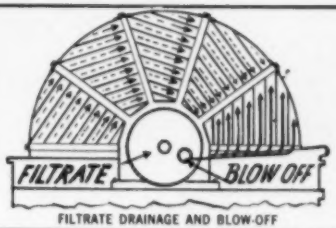
(a) Filtering rate of material.
(b) Size of filter required.
(c) Type of filtering media needed.
(d) Optimum speed.
(e) Moisture expected in cake.
(f) Vacuum equipment required.

We are anxious to work with your metallurgists.

WRITE US TODAY

PATENTED DESIGN

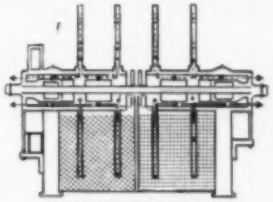
Patented grooves in filter segment drain direct to "suction and blow-off tube" which extends along leading edge of segment. This design increases suction efficiency... collects thicker, more uniform cake... means positive drainage just ***before*** "blow-off" and discharge of cake.



FILTRATE DRAINAGE AND BLOW-OFF

VALVES. . . . . .

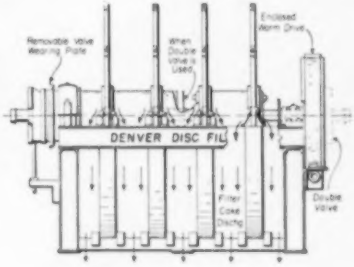
Double valves and divided tank can be provided... gives flexibility where two products are being treated. Use of two outlets permits separation of filtrates (strong solution from weak solution) where washing cycle is used... gives higher flow capacities if desired.



CROSS-SECTION FROM REAR

CONSTRUCTION. .

Wearing parts are protected and easily replaced. Bearings are protected from muck and dirt. Tank and frame are double-welded steel construction. Filter is compact... gives greater filter area in less floor-space, less head room.



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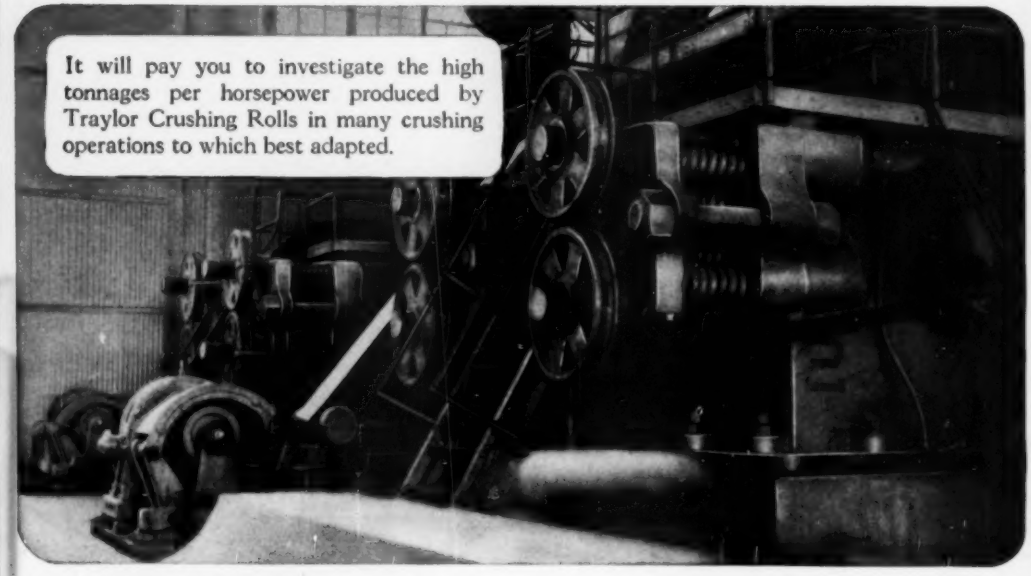
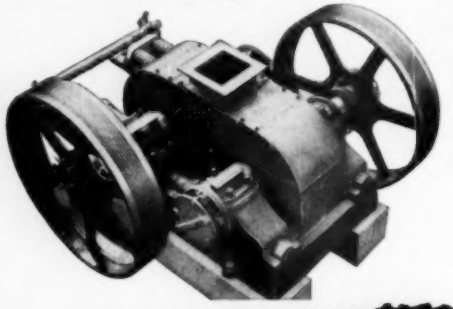
Great Producers

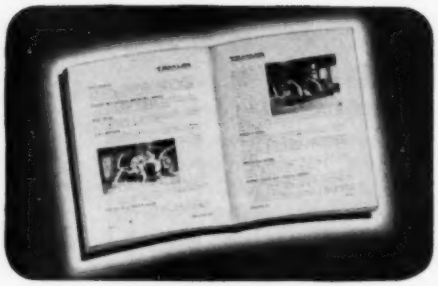
. . . with Low Power Requirements



It will pay you to investigate the high tonnages per horsepower produced by Traylor Crushing Rolls in many crushing operations to which best adapted.

Here are actual performance figures - - Traylor 78" x 20" Rolls (main illustration) operated by a large mining company, consistently crush primary feed of 310 t.p.h. plus 180% circulating load to 52% minus 8 mesh. And have done so for a long time. A pair of smaller rolls, 36" x 16" and 42" x 16" consistently runs 125-150 t.p.h. to ⅛"-3/16". They then reduce oversize, which has been screened out, to 6 mesh at a rate of 120 t.p.h.





Traylor Crushing Rolls are made in three types and thirty-three sizes to meet the requirements of all operations. Bulletin 4637 contains descriptions and specifications of all types and sizes. Write for your copy today.

Traylor

Rotary Kilns, Coolers and Dryers
Grinding Mills . Crushing Rolls
Jaw, Reduction and Gyratory Crushers

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Loading operations at one of the huge vats of Sulphur at our Newgulf, Texas mine. Such mountains of Sulphur are constantly being built at our mines, from which shipments are continually made.

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CONTINUOUS VACUUM

PARAMOUNT IN EFFICIENCY AND MECHANICAL DEPENDABILITY

Dependable for Continuous Duty — Fully Automatic. Morse Drum and Disc Filters are highly regarded for satisfactory performance and low maintenance—made in a wide range of sizes to meet most all requirements.

Morse Drum Filters are particularly adapted for flotation concentrate, cyanide slimes and other dewatering problems that require wash-water apparatus.

Morse 3' Drum Filters, of all metal construction, are ideal for many applications where only small or medium capacity is necessary.

Morse Disc Filters are especially adapted for filtering more than one character of concentrate or material where separate filtrates are desired.

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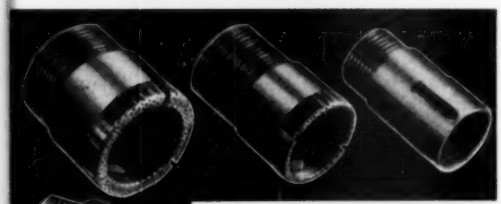
Established 1898

DENVER, COLORADO, U.S.A.

(Cable Morse)



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Mining Engineer familiar with foreign ores and minerals, for export-import house. Location, New York, N. Y. Y3106.

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Geologist with about two years experience in the field on exploration and report writing for work in British Guiana. Salary, $4200-$4800 a year. Y3140.

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A. Let Gardner-Denver GD14 Mine Car Loaders handle the mucking. That's one good way to speed up the drilling-blasting-loading cycle in every heading. With these fast, clean-mucking loaders, you can schedule the work to keep every loader busy throughout every shift. You'll find that productivity and efficiency are increased — mining costs are lowered.

GD14 Loaders give you full-time availability — as proved by their performance in hundreds of mining installations. Gardner-Denver *master engineering* features provide speed for the job and safety for the man. Get all the facts — send today for Bulletin MCL.

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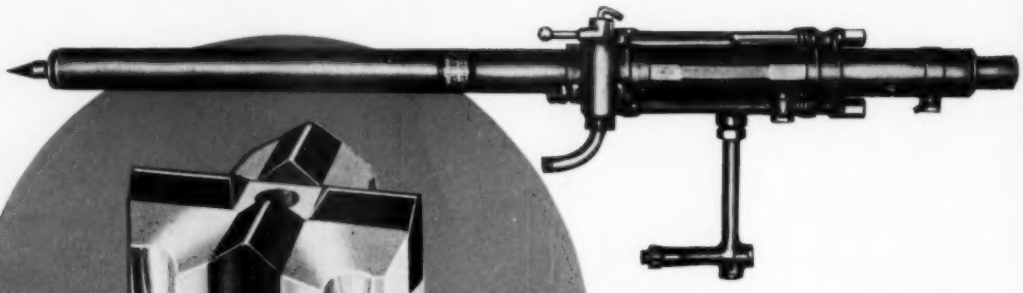
Three JOY T-350 Drifters on the job. These drills are available with either hand or Pistonmotor feed, as desired.



COMPLETE LINE OF STOPERS

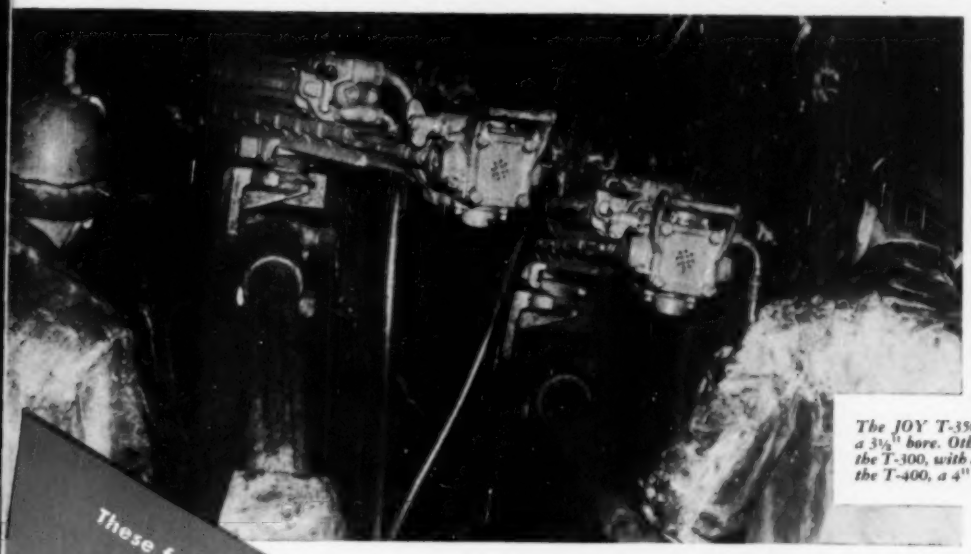
Above: the JOY "Silver Streak" S-91 Stoper, equipped with "thumb-flip" safety rotation control and the patented JOY Dual Valve. Four other models include the exclusive JOY "Safety" Stoper, for drilling in dangerous raises.

SULMET BITS HELP TO CUT COSTS

Left: JOY Sulmet Bits, tipped with tungsten carbide, assure faster drilling speed and longer feeds in hard rock, and permit fewer bit changes. Less loss of gauge means that deep holes can be bottomed in many cases with one size bit.

You can drill a Round *FASTER*—
at less cost per foot with

JOY *SILVER STREAK* DRIFTERS



The JOY T-350 Drifter has a 3½" bore. Other models are the T-300, with a 3" bore; and the T-400, a 4" drifter.

These features mean savings for you in Drilling Speed, Service Life, Air Consumption and Maintenance!

- exclusive CADMIUM PLATING, inside and out, for easier "running-in" and protection from rust.
- patented DUAL VALVE—makes air do more work, gives cushion control and positive, powerful piston action.
- exclusive LOCKING CHUCK, with 30% greater bearing area for longer life of chuck parts and drill steel.
- patented PISTONMOTOR FEED, for smooth steady advance and safe retraction.
- automatic DRILL STEEL CENTRALIZER, for accurate, safe and easy spotting of holes.

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W&D M2095

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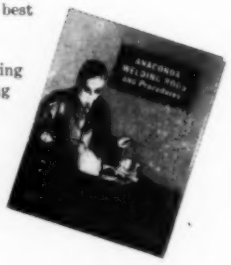

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You can depend on ANACONDA BRONZE WELDING RODS

Manufacturers News

Equipment

Foote Bros. Gear and Machine Corp. of Chicago and The Louis Allis Co. of Milwaukee have announced a joint program for the manufacture and sale of a complete line of gearmotors. The new Foote Bros.-Louis Allis gearmotor makes use of hard helical gears and other moving parts which have been processed and heat treated under new and improved methods of manufacturing control. These methods are said to have produced new high standards of performance with regard to load-carrying capacity, wear life, quietness of operation, and compactness of design.

A new laboratory unit, employing the patented Jeffrey-Steffensen counter-flow principle of magnetic separation, has been developed for the treatment of finely ground magnetic iron ore by The Jeffrey Mfg. Co., Columbus, Ohio. It is stated that these separators are being used in the treatment of taconite in Minnesota, and on eastern magnetite. They are also used for the recovery of magnetic media in the heavy-media processes.

Marlow Pumps, Ridgewood, N. J., has announced the addition of a new line of straight centrifugal pumps. This expands the full Marlow line to 56 basic series of pumps, a line considered by Marlow engineers to be the largest of its kind in the world and one which can meet most industrial pumping requirements.

A new plan under which customers may have their choice of engine when ordering the C Roadster Tournapull has been announced by R. G. LeTourneau, Inc., Peoria, Ill., manufacturer of heavy earth-moving equipment. The Tournapull is available with either a General Motors 6-71, Cummins HBI-600, or Buda 6-DC-844 engine. It can be used as a prime-mover for a 16-ton Carryall Scraper; a Tournarocker, LeTourneau's rear dump rock wagon; or the Tournahopper, their bottom-dump hauling unit. The C Roadster now is equipped with a heavy-duty five-speed transmission. Top road speed is over 30 mph.

The Davey Compressor Co. of Kent, Ohio, has a new line of custom-built, heavy-duty compressors, known as Air Chief AIRistocrats. Production of these units will be limited to 100 per year in 105 and 210 cfm sizes, and 50 per year in 160 and 315 cfm capacities, according to Paul H. Davey, president. AIRistocrat features include an extra heavy-welded steel vibrationless frame, electric starting, fenders, locking gas cap, hand parking brakes, and electric service brakes. Model 105, shown in the illustration, delivers 105 cfm at 100 lb pressure. It is 124 in. long, 66 in. wide, and 68 in. high. Weight is 4100 lbs.

The new 44-ton side-dump trailer train, (see cut) built by Easton Car & Construction Co. of Easton, Pa., will be used to haul cement rock on a half-mile round trip between shovel and crusher at the Marquette Cement Mfg. Company's quarry in Oglesby, Ill. The train combines two large capacity semitrailers, coupled by means of a pneumatic-tired dolly. Each trailer is equipped with a drop-door, side-dump body of 22-ton capacity. An outstanding feature of the body design is the patented automatic down-folding side door, which drops open as the body is raised for dumping. Two hydraulic, three-sleeve, single acting, telescopic hoists, operated from the driver's cab, are used to raise each body. The trailers are equipped with Bendix-Westinghouse air brakes.

Easton Car & Construction Co.'s 44-ton trailer train.

Bulletins

Roof Bolting. The Joy Mfg. Co., Pittsburgh, Pa., has released a new 12-page bulletin, M-206, on "Roof Bolting with Joy Stopers and Mine-Air Compressors." The bulletin describes the equipment required and the methods used for this latest system of roof support. Roof bolting in coal mines not only gives greater protection and safety, but it is also an aid to production.

Wire Rope. Stainless steel and Monel metal wire ropes are becoming more popular where corrosion affects uncoated carbon steel and galvanized ropes, according to the manufacturer, the Macwhyte Co., Kenosha, Wis. Folder 49-30 catalogs 133 different sizes and types of wire ropes.

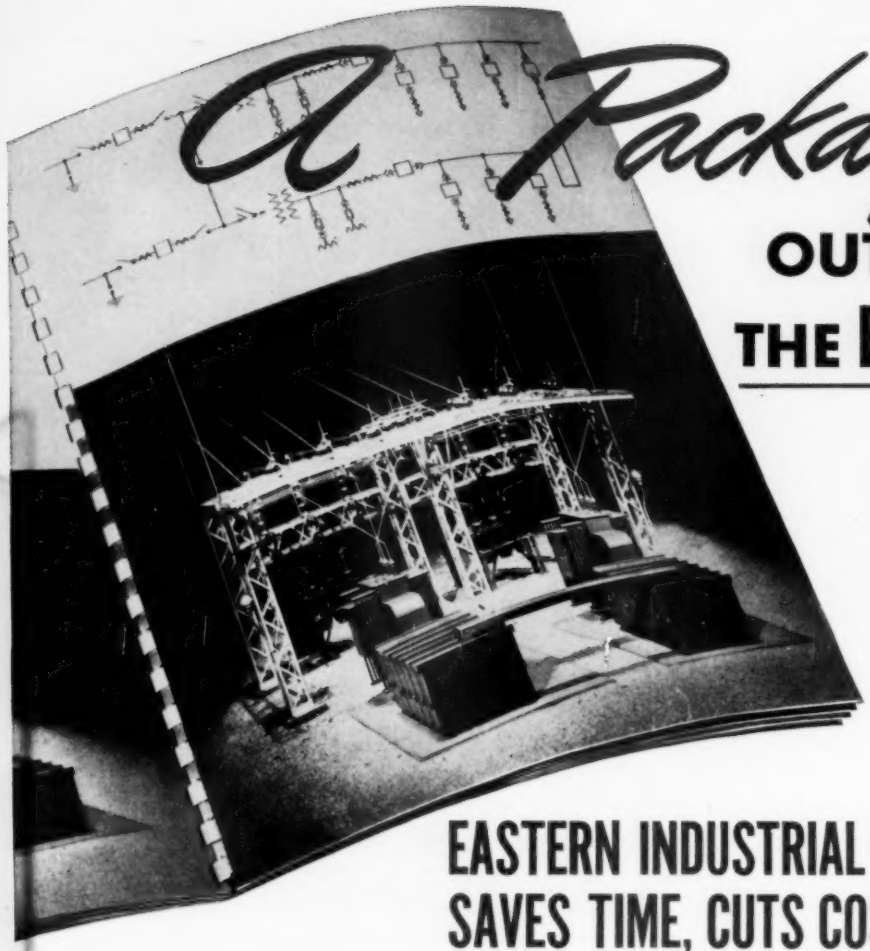
Conveyors. The Jeffrey Mfg. Co., pioneers in the field with its Jeffrey-Traylor line of electric vibrating equipment, has announced the development of the Jeffrey MV Conveyor. Bulletin No. 826 describes this mechanical vibrating conveyor.

Air Conditioning. Bulletin No. 112, issued by the Niagara Blower Co., New York, describes their controlled-humidity method of air conditioning.

Crushing and Screening. An interesting engineering brochure, containing many drawings with suggestions for arranging equipment to make up complete portable and semiportable crushing and screening plants, is available without charge on request to the Crusher Division, Nordberg Mfg. Co., Milwaukee.

Spiral Concentrator. Bulletin No. 10, "Humphreys Spiral Concentrator Closed Circuit Test Unit, Manual of Operating Instructions," has just been issued by The Humphreys Investment Co. It describes the Humphreys spiral concentrator, why it works, its field of application, procedure for test work, and actual operation, giving the proper conditions for obtaining optimum results.

Rubber Transmission Belting. A new 4-page catalog section on transmission belting has been issued by The B. F. Goodrich Co. of Akron. Various types of belting are pictured and described; recommended uses are listed and specifications given.



A Package OUT OF THE BOOK

EASTERN INDUSTRIAL PLANT SAVES TIME, CUTS COSTS,

BY ORDERING COMPLETE OUTDOOR SUBSTATION *As a Unit*

A *complete* package substation, plus *complete* factory-assembled load-center units for indoor distribution, gave this new plant a power system having the two vital requirements they could get in no other way.

Quick, easy ordering and installation

Only one set of specifications and one order were needed for *all* the equipment comprising the package substation. The same was true for the load-center units. The long delay and high cost of ordering the substation components piecemeal were eliminated. Installation was made simple and easy, too—the substations were shipped in factory-assembled sections as large as shipping facilities permitted.

Exceptional assurance of power continuity

This plant's operations are such that a power interruption would likely be extremely costly. Unusual precautions were warranted to assure power reliability. The answer—the best and the most economical answer—was in the system shown at the far right, incorporating the full design co-ordination and metal-enclosed safety of modern General Electric substations.

When you consider any substation application, be sure to look into General Electric's *unit approach*. We feel you'll be pleasantly surprised, both by the initial economy and, later, by the on-the-job advantages of increased flexibility and safety. Apparatus Dept., General Electric Company, Schenectady 5, N. Y.

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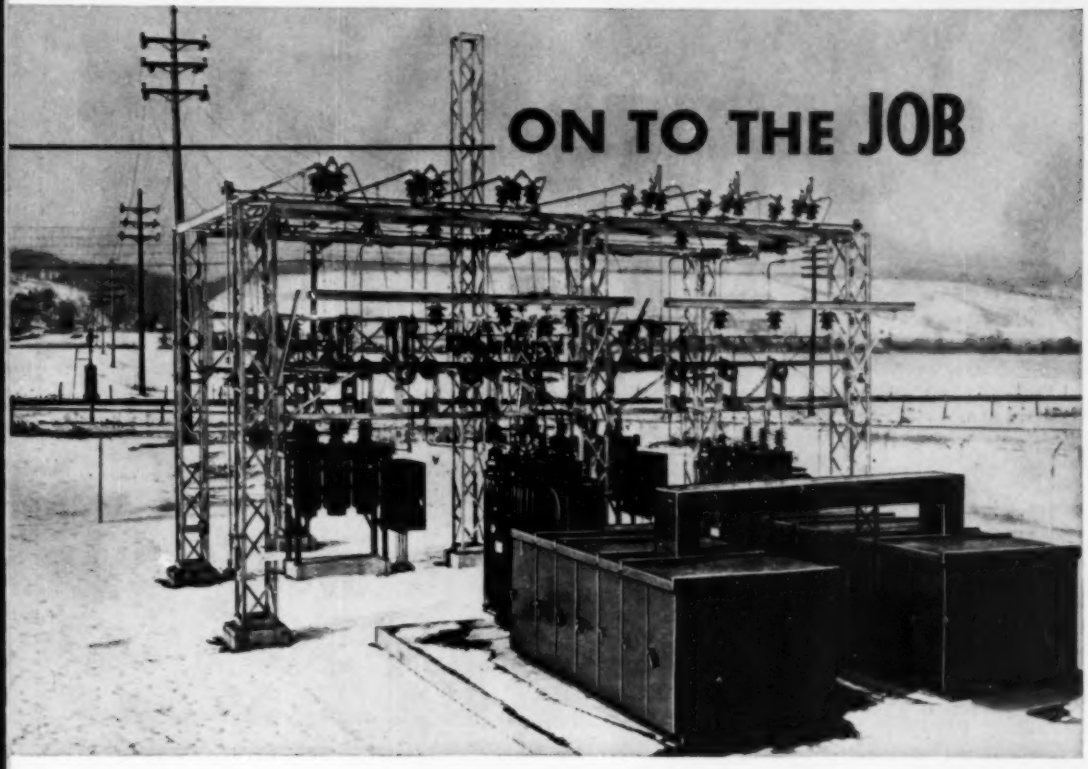

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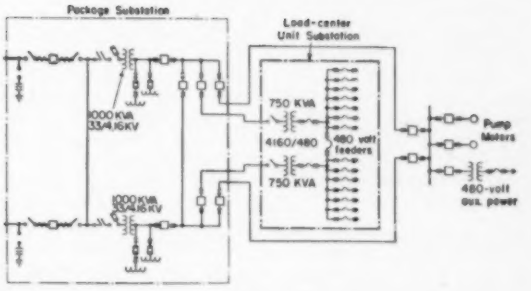
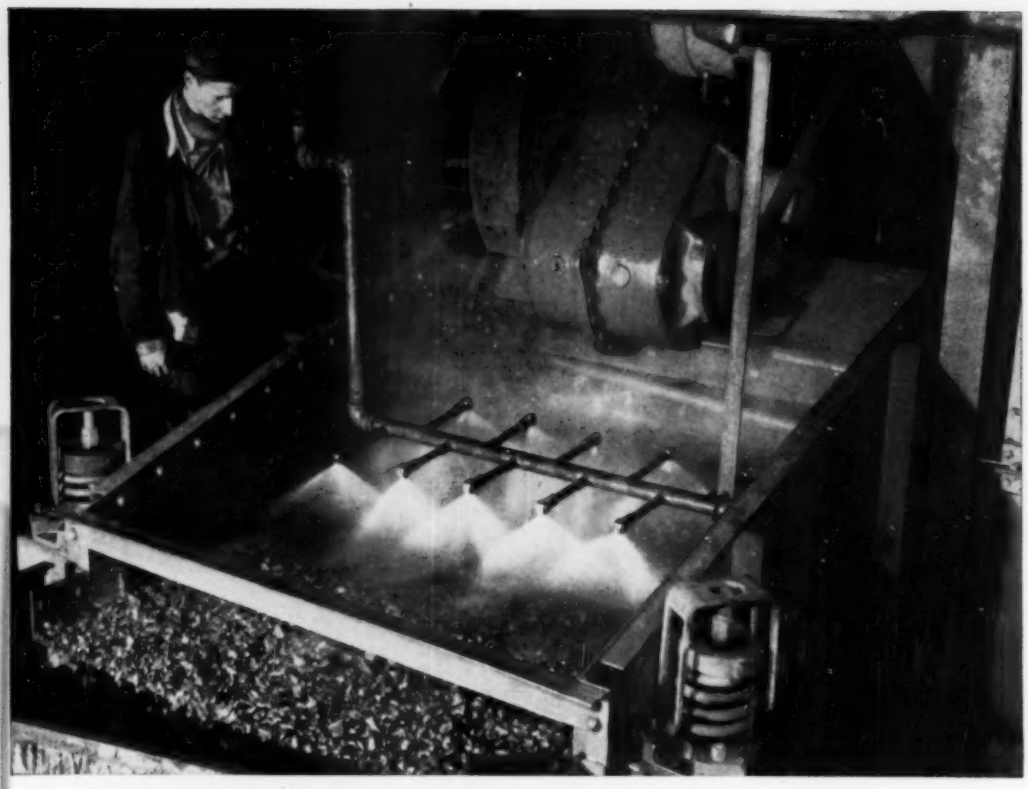
The General Electric *package substation* extends the economical, easy-to-order, easy-to-install features of load-center unit substations to the full range of outdoor substations.

This means you can buy a complete substation *as a unit*—whether it be a 100-kva load-center unit or a complete outdoor substation like that above, with ratings up to 40,000 kva.

You'll find helpful information on all G-E substations in GEA-3592 (Load-center unit substations), GEA-3800 (Master unit substations), and GEA-4500 (Package substations). Get copies from your G-E sales representative, or write to Apparatus Dept., General Electric Company, Schenectady 5, N. Y.



GENERAL ELECTRIC



Top Results in Media Recovery — with *Low-Head* Washing Screen

THE FIRST SECTION of this *Low-Head* vibrating screen drains media from iron ore tailings in a Minnesota heavy density plant. The remainder of this single deck, 5x16-ft screen washes media from tailings. Operators are highly pleased with the high media recovery.

Low-Head screens are widely used for draining or washing media from ore in heavy density applications . . . for screening log washer products . . . for wet or dry sizing, rinsing and dewatering semi-fine to semi-coarse ores and aggregates.

- Horizontal operation saves headroom, reduces installation costs.
- Entire screen body "stress-relieved" to eliminate strains around welds.
- Can be suspended by cables or floor mounted on rubber mountings.
- Available with screen surfaces for abrasive ores.

Allis-Chalmers builds six separate types of vibrating screens — a complete line for every application from heavy duty scalping of run-of-mine ore to electromagnetic screens for making fine separations down to 65-mesh (wet).

There is an Allis-Chalmers representative near you who will gladly look over your operations, show you how you may be able to reduce your operating costs with proper screening equipment.

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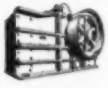


Jaw Crushers

Kilns, Coolers, Dryers

Gyratory Crushers

Letters to the Editor

Hospitality and AIME Meetings

Before me is Mining Engineering, November 1949—opened to page 27. In my estimation you have hit upon a vital point with respect to our mining industry and its attitude toward the AIME and its meetings. Too many mining companies fail to realize that men in the lower echelons of their technical staff are the ones who can contribute the "new blood" to our technical meetings and at the same time derive the most from the broadening atmosphere of these meetings. Attendance is too frequently reserved for the "top brass".

There is, however, an equally vital point which is not mentioned in your editorial. Perhaps the reason that we always see the "old crowd" at meetings of the Institute is that too little attention is paid by the older and more established members to the less-acquainted new members and visitors (potential AIME timber in many cases).

More than half of the participants in the technical sessions of the Industrial Minerals Division at Columbus, Ohio, last fall were non-Institute men. Having guided the arrangements for this program, I felt responsible for the welfare of these men at the meeting. Though I was unable to attend, I wrote a number of the leaders in the affairs of the Division requesting that these men be made especially welcome and made to feel a part of the group—socially as well as technically.

No feeling is more disconcerting than to attend a technical meeting in the company of strangers. A person can take but little away from such a meeting. It may sound like evangelism, but great good would result if every established Institute member would seek out and cultivate the acquaintance of several younger members or potential members at each meeting. The advantages would be mutual and the net result would be a stronger AIME.

In closing I would like to express my approval of your recently adopted format for the Mining Engineering.

Robert C. Stephenson, Chairman
Mining Geology

Concentration at Coronet

In reporting on the visit to the Eleanor Mine of the Coronet Phosphate Company, in the December 1949 issue of Mining Engineering, the statement is made that table agglomeration is used for concentrating the coarser sizes of the phosphate. Our coarser sizes, those ranging between 14 and 35 mesh roughly, are treated on belt concentrators rather than tables. A description of these machines will be found in the November 1948 issue of *Rock Products*, with a complete discussion of the Eleanor Plant.

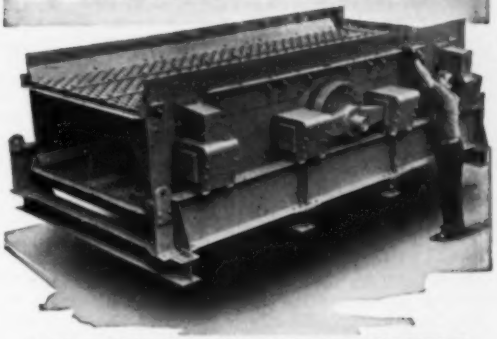
This method of concentration was developed by the Coronet Phospate Company some years ago and so far as we know, they have not been used anywhere else to date. One of the other phosphate miners in Florida is installing these machines at a new operation which will start up shortly.

In addition to Humphrey Spirals, tables and belt machines, a fourth method of handling these coarser sizes is in use in the Florida field, namely, the underwater screening of the oiled phosphatic feed.

We would also comment that the sizing split between our belt feed and the flotation feed is made in General American Hydrosizers rather than in Fahrenwald sizers.

J. R. Sheffield, President
Coronet Phosphate Co.

TY-ROCK—*the Ideal Screen*
for HEAVY LOADS—COARSE MATERIALS!



USED THROUGHOUT THE WORLD WHEREVER LARGE TONNAGES OF ROCK OR ORE ARE SCREENED!

Manufacturers of Woven Wire Screens and Screening Machinery

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Space Reservation — 1st Preceding Month
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Magazine Issued — 23rd Preceding Month

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Authors in This Issue

Gilbert H. Cady (p. 275), is senior geologist and head of the coal division of the Illinois State Geological Survey. Born in Chicago, he attended Lewis Institute there, then Northwestern, Yale, and the University of Chicago, earning A.B., M.S. and Ph.D. degrees. He joined the ISGS in 1907, after two years of teaching at Southwestern College in Kansas. A year later he was back at Southwestern, and then went on to teach at both Wesleyan and Northwestern Universities. In 1919 he worked as a geologist in Yunnan, China, and then for six years headed the department of geology at the University of Kansas. Dr. Cady is an AIME Member, and has made numerous contributions to AIME Transactions volumes.

E. H. Crabtree

E. H. Crabtree, Jr. (p. 217), director of milling for the Eagle-Picher Mining & Smelting Co., Miami, Okla., has been published by the AIME on four other occasions. He was born in Lewiston, Mont., in 1904, attended Main Ave. High School in San Antonio, Texas, and went to the Colorado School of Mines, graduating with an E.M. in 1927. Eight years later he became mill superintendent for Eagle-Picher, and in 1940 became a metallurgist with the Arizona Bureau of Mines. Eagle-Picher took him back as chief metallurgist in 1943, and four years later he took over the job he now holds. Mr. Crabtree is an AIME member.

W. W. Anderson (p. 256) of Evanston, Ill. is technical director for the western division of the Commercial Testing and Engineering Co. in Chicago. Born in Kenosha, Wisconsin, he attended Evanston Township H.S., and then took a B.S. in mining engineering from the University of Illinois in 1930, at which time he joined the above named firm. Mr. Anderson is a member of the Institute. . . . **C. Kremer Bain** (p. 201) has been with the St. Joseph Lead Co. at Bonne Terre, Mo., since 1920, and is now a consulting engineer for that organization and for the Soc. des Mines de Zellidja. Born in Louisville, Ky., he attended Lexington High School in Lexington, Ky., and took his engineer of mines degree from the University of Kentucky. Mr. Bain spent the first four months of 1949 in French Morocco as a consulting engineer for the Soc. des Mines Zellidja. He is an AIME member, lives in Bonne Terre, and enjoys golf in his spare time.

Leslie C. Richards (p. 285) is now engaged in his own private practice in Portland, Oregon. He has worked at the Sunshine Mine, Kellogg, Idaho, and at various gold properties in Montana, Idaho, Washington and Oregon. Born in Tacoma, Washington, he attended Glendale High School, Glendale, Calif., and has a B.S. from Oregon State College. During the war Mr. Richards was a Navy lieutenant, serving in the south and central Pacific areas. His chief interest lies in industrial minerals, but he likes to relax in the Pacific Northwest's many ski areas, particularly Sun Valley. Mr. Richards is also an AIME member. . . . **Richard Hamburger** (p. 207) hails from Globe, Ariz., where he's a junior engineer for the Inspiration Consolidated Copper Co. He attended the Park School in Baltimore, Md., Johns Hopkins Univ., and then the Univ. of Michigan, where he received an A.B. in geology. Mr. Hamburger is an Institute member, and this is the first paper of his to be published by the AIME.

S. B. Tuwiner

S. B. Tuwiner (p. 226), co-author of "Effect of Conditioning on Flotation of Chalcocite" with S. Korman, is assistant director of research for the Phelps Dodge Copper Corp., Laurel Hill, N. Y. Mr. Tuwiner first joined Phelps-Dodge in 1936. Mr. Tuwiner lives in Baldwin, L. I., N. Y., and is a member of AIME. When not researching, he enjoys gardening—and practical jokes.

H. A. Baumann (p. 265) is another AIME member. He has spent several years in various capacities in the coal industry, from foreman to plant superintendent. He has also worked at combustion engineering research on single retort, industrial and domestic stokers. Now he is air conditioning and refrigeration engineer for the National Mine Service Co., Whiteman Div., Indiana, Pa. Born in Noblestown, Pa., Mr. Baumann attended Findley Vocational School, took his B.S. from the Univ. of Pittsburgh. He has presented one other paper before the AIME. His "off-duty" pleasures include golf and fishing.

J. F. Poland (p. 279) of Sacramento, Calif., is a geologist with an A.B. from Harvard and an M.A. from Stanford Univ. He is assistant geologist to the district geologist with the ground water branch of the USGS in California, and has held that post for the past ten years. He has been a geologist with the Tropical Oil Co. in Colombia, S.A., and spent eight years as a student - instructor at Stanford Univ. He was born in Boston and attended Concord, Mass., High School. This is his first paper to be published by the AIME. Mr. Poland enjoys relaxation on the tennis courts and a bit of spare time fishing.

R. A. Kinzie, Jr. (p. 289) is an AIME member, and lives in Santa Cruz, Calif. He is chief chemist and superintendent for the Santa Cruz Portland Cement Co., in Davenport, Calif. Previously he had worked in the oil fields for the General Petroleum Corp. Mr. Kinzie was born in Treadwell, Alaska, attended St. Ignatius High School in San Francisco, and then went to the Univ. of California for his B.S. degree. He has presented two other papers before the Institute. His chief pleasure, says Mr. Kinzie, is "living in Santa Cruz."

H. N. Eavenson (p. 193) is extremely well qualified to outline for us the present position of the bituminous coal industry. He has been a consulting civil and mining engineer for almost thirty years, and now heads his own firm, Eavenson and Auchmuty, in Pittsburgh. He was President of the AIME in 1934, a Vice-President in 1932-3, a Director from 1917-20 and 1931-33, and headed the Coal Division in 1930-31. When Mr. Eavenson became a consultant in 1920, he brought to his work more than 28 years experience in the coal industry. He has a B.S. and C.E. from Swarthmore, and has found time to author numerous papers for AIME Transactions.

S. S. Arentz

S. S. Arentz (p. 204) is general superintendent for the Combined Metals Reduction Co., Pioche, Nev. He has a B.S. from the Mackay School of Mines, Univ. of Nevada, and has worked in Idaho, Colorado, and on the Hoover Dam site. Mr. Arentz is an AIME member. He enjoys taking active part in community activities, and is a regent of the Univ. of Nevada.

For greater metallurgical capacity – *specify* FAGERGREN

The Fagergren Flotation Machine handles *more* tonnage with *less* power consumption . . .

HIGHER SPEED OF FLOAT is accomplished by more intense aeration and shallower froth column. Hence, treatment time is reduced (by as much as 50%) and more *tonnage per hour* results.

GREATER RECOVERY of valuable mineral is assured by increased bubble formation and the thorough diffusion of finer bubbles. Alternate compression and expansion of the pulp as it passes through the rotor-stator causes intense aeration and agitation. This action provides more interfaces, and is essential to rapid, efficient mineral separation. The result is improved metallurgical efficiency.

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THE FAGERGREN FLOTATION MACHINE, an exclusive WEMCO product, assures operators of *higher production* per cubic foot of cell volume at *lower cost.*

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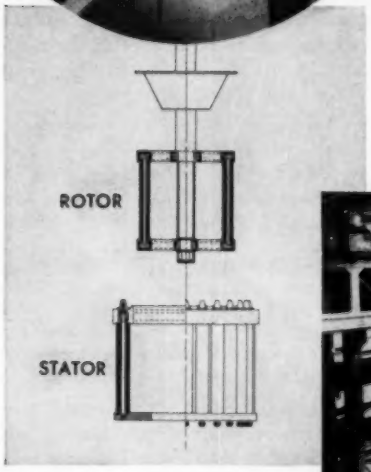

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NI-RESIST...
AN ECONOMICAL METAL

for resisting Corrosion, Erosion and Metal-to-Metal Wear

NI-RESIST* is the trade name of a high nickel cast iron – one of the most economical corrosion-resistant engineering materials ever developed.

NI-RESIST, mechanically similar to gray iron, and resembling austenitic stainless steel in many characteristics, provides a unique combination of properties at moderate cost . . .

RESISTS CORROSION

NI-RESIST resists corrosive attacks of *acids, alkalies* and *salts* to a degree unmatched by any other product of the cast iron industry. In strong corrosives it has 20 to 200 times the resistance of plain iron, and 5 to 50 times the resistance in mild environments.

CURBS WEAR

NI-RESIST reduces wear and galling, because graphite particles are distributed throughout its structure, as in gray iron. From this, and its work-hardening characteristics, come the superiority of NI-RESIST castings for components in metal-to-metal wear service.

CHECKS EROSION

This superior wear-resistance . . . together with inherent resistance to corrosion . . . makes NI-RESIST an outstanding material for curbing erosion by liquids and slurries. By long, trouble-free control of erosion, NI-RESIST castings reduce maintenance costs, production tie-ups and the expense of replacements.

WITHSTANDS HEAT

Although primarily used to resist corrosion, erosion and metal-to-metal wear . . . NI-RESIST is also specified for resistance to elevated temperature effects. Castings of NI-RESIST show up to 10 times better scaling resistance, and up to 12 times better growth resistance than those of plain iron at 1300 to 1500° F.

MACHINABILITY and OTHER PROPERTIES

NI-RESIST of normal hardness machines like 200 BHN gray iron and is readily weldable.

NI-RESIST has high specific electrical resistance. Thermal expansion can be controlled, from 60 per cent higher than that of plain iron to a low approximating that of Invar.

NI-RESIST is usually lower in cost than most other corrosion-resistant alloys. It is produced by authorized foundries only, in all industrial centers of the country. NI-RESIST castings have no more limitations in size and complexity than those of any gray iron.

APPLICATIONS

Several types of NI-RESIST *are available. All provide the fundamental properties described above, and differ only in certain special characteristics to meet a variety of industrial demands.*

Applications include: Salt filter drums, oil refinery tube supports and headers, turbine nozzle rings, cylinder liners, valves and fittings, furnace rollers, textile rolls, comminuter parts, blow pit pipes, precision machine tool spindle heads, bridge, and work supports, magnet housings, sugar retorts, etc.

FULL INFORMATION

May we send you two booklets? One, entitled, "Engineering Properties and Applications of NI-RESIST," includes corrosion data on NI-RESIST and cast iron under 400 different corrosive conditions. The other, entitled, "Buyers' Guide for NI-RESIST Castings," lists producers of NI-RESIST castings. Both are yours for the asking. Write for them today.

*Reg. U. S. Pat. Off.

EMBLEM OF INCO SERVICE

THE INTERNATIONAL NICKEL COMPANY, INC. 67 WALL STREET NEW YORK 5, N.Y.

* The greatest and richest iron ore body in the world was discovered by U. S. Steel in Venezuela in April 1947, and is disclosed for the first time in this issue (p. 178). One solid mountain of ore, Cerro Bolivar, will furnish more tonnage than the famous Hull-Rust pit at Superior has, or can produce. Total resources are on the order of 2 billion tons plus.

Big Steel is now moving with great speed on construction to bring out this ore, and 10 million tons are scheduled for shipment into Baltimore and Birmingham by late 1952. This ore will arrive at Pittsburgh at a price lower than present natural ores from Superior. The Venezuelan properties will be completely mechanized.

* U. S. Steel and independant companies are embarking on a taconite beneficiation program of tremendous proportions, which must be speeded to completion in the 1950's (see p. 185).

By 1960 over $1.2 billion dollars will be spent on Venezuela, taconite, and Labrador to solve the United States iron ore crisis. In addition, U. S. Steel will spend another $300 million plus for a new steel mill at Trenton, N. J., before 1955.

* The Deister Concentrator Co. gave $500 to the 1950 Coal Division Scholarship Fund.

* Howard N. Eavenson, in analyzing the bituminous coal situation (see p. 193), suggests that anti-trust laws be extended to labor unions, disinterested trustees to administer pension funds, begin pension funds now and stop the collection of all "check-offs" for union dues or funds.

* Minerals and metals worth 13.9 million dollars were produced in the United States in 1949, Secretary of the Interior Oscar L. Chapman announced. Although this value was 11 pct less than in 1948, it was the second greatest in history. Nonmetallics was the only major group that in 1949 equaled the 1948 production high. Mineral fuels decreased 12 pct and metals 15 pct.

* Prospects for making gas from coal are seen to be unattractive by the Pittsburgh Consolidated Coal Co. because the success of operation depends on price and supply of coal which presently are not attractive. The company has shut down its gasification plant at Library, Pa.

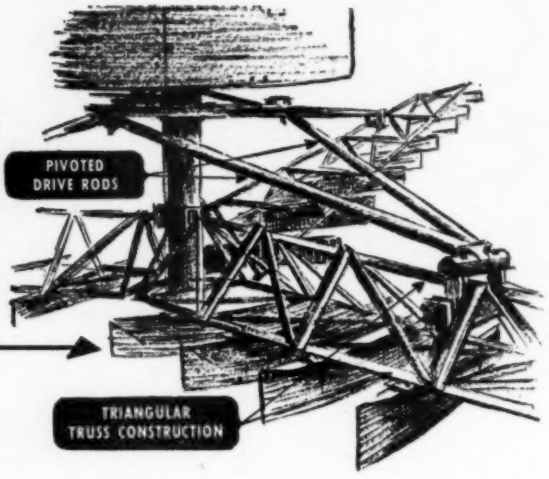
* Economic prospects for the immediate future in the United States are good and business probably will continue at its present high levels throughout at least the early part of 1950, Secretary of Commerce Charles Sawyer said in his year-end report to the White House based on his personal survey of business conditions.

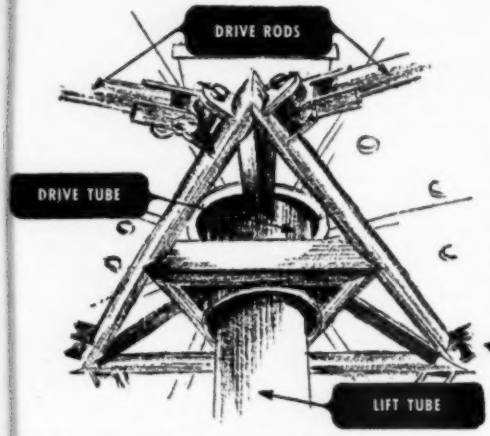
* A 467-ft well 34 miles north of Rock Springs, southwestern Wyoming, is said to be flowing a 5 pct solution of nearly pure trona (sodium carbonate) at the rate of 10,000 barrels a day. Trona, natural soda ash, is mined for the chemical trade at Searles Lake, Calif., and from about 1500 ft underground near Green River, Wyo., and has been in short supply.

* The AEC is "very much interested" in a discovery of radioactive material that has been made in Baraga County bordering Lake Superior in the Upper Peninsula of Michigan, reported F. G. Pardee, Michigan State mining engineer. The material has been identified as uranium oxide in the form of pitchblende or uraninite. The discovery, made by a geologist of the Jones & Laughlin Ore Co., is the only instance in which the commission has certified samples of Michigan rock.

Here's why DORR TYPE H Hydroseparators and Thickeners can take tough, heavy-duty operation . . .

Key to Type H strength and rigidity is triangular construction . . . the basic structural principle of the three-legged stool. Note that the Type H has three rake arms . . . each constructed of triangular trusses. Note, too, the two pivoted drive rods for each arm and how they too form triangles with the center truss as shown in the sketch at the right.

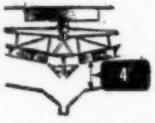




In the closeup sketch at left, the series of triangular trusses connecting the drive rods to the central drive tube are clearly shown. *All* structural supporting members form triangles . . . providing a pattern of rigid strength to withstand heavy torque loads.

Powered by a heavy-duty balanced spur-gear drive, Dorr Type H Hydroseparators and Thickeners are now available in sizes ranging from 50 to 100 feet in diameter. Single units will handle 100 to 1,000 tons of solids per hour. May we give you further information?

PLUS THESE IMPORTANT FEATURES . .



AUTOMATIC HYDRAULIC LIFT
For overload protection, an automatic hydraulic lifting device is provided, activated by the torsional load on the mechanism. Lift tube and rakes raise quickly . . . up to four feet—when an overload is encountered . . . lower slowly as the overload is reduced.

FREE, CENTER CONE DISCHARGE
Entire rake and drive mechanism is supported by a truss spanning the tank. Result . . . free, center cone discharge . . . no impaction of heavy solids in the discharge cone.

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RESEARCH ENGINEERING EQUIPMENT

Something Left to Be Done

PRESIDENT Truman's economic report to Congress calls for an increase of 40 billion dollars in the value of goods and services produced in the United States by 1955; this is an increase of 16 percent over our 1949 production. If such an increase is to be realized, it will be possible only by a comparable increase in the production of steel—the keystone of our economy. Going one step further, since the raw materials of steel are iron ore, limestone, and coal, the President's goal can be reached only through the efforts of the mining industry. Coal and limestone reserves are not critical at this time, but the abundance of iron ore reserves has been subject to controversy. In this issue we proudly present "Cerro Bolivar—Saga of a Crisis in Iron Ore Averted" (a story which has been seven years in the making and three years in the getting).

It is evident that, roughly two years from now, our iron ore will begin to come from present high-grade producing areas, from foreign sources, and from Taconites. The immediate problem facing the mining man is twofold —engineering and political.

Engineeringwise we are equal to the task. We will unravel the beneficiation problem of low-grade ores; we will mechanize our mines; and we will be supplied with the huge ore vessels necessary to bring the ores to our shores. This will be done as described in the following pages so as not to increase the price of finished steel by more than one dollar.

Politically, the problem is not so simple. Strangely enough, the very Presidental administration that makes these prophesies of increased production for the next five years with even greater things hinted beyond this, has not aided to the fullest extent the basic industry that will make this possible. Our biggest foreign iron developments are, so far, being financed by private capital. The sum which will be needed by 1965 will be of the magnitude of 1.2 billion dollars—an awe-inspiring figure.

The bigness which gives the strength for United States Steel to undertake the vast exploration program, which was needed to find Cerro Bolivar and now to transport its treasure to our shores, is under attack by the administration as evidenced by the suits against Du Pont and other corporations. Venture capital to develop the Labrador deposits being spearheaded in this country by M. A. Hanna faces discouragement in heavy corporation taxes. Increases in wages obtained under pressure are freely allowed as the government stands by, but the steel price increase by the Corporation is being investigated by the O'Mahoney congressional committee.

Sometimes it is difficult to believe that we are all working for the good of America. The people of the United States cannot receive the increased income the President envisages unless productive work is performed. Productive work means jobs and they require capital expenditure. The corporations are the voice of the stockholders—who number equally with the wage earners—and unless the corporation brand of politics is stepped up to the tempo of the unions, America may suffer from the policies of those who would aid her but in actual practice don't know how.

It's Everyone's Business

JAN. 17—In what appears to be a general spirit of post-Christmas emotional malaise, most adult Americans have bidden farewell to the Forties and turned with no perceptible enthusiasm toward the Fifties. The dying decade had started amid a phoney war which verged on the romantic, stirred the pulse with excitement and seemed far away. A phoney peace dominates the new decade. The romance is no longer there. Nor is the excitement. Nor is the distance.

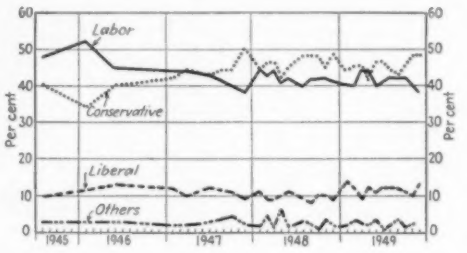
While most of the world preferred to be chronologically correct and hold off for another year before celebrating the demise of the first half of the twentieth century, the United States with true competitive enterprise and a great whirl of words scurried into the second half a year early. This undue haste might perhaps have been dictated unconsciously by a yearning for the pattern of history to be repeated. For many centuries, ever since Roman times, the second half of most centuries tended to be far more tranquil than the first half.

In Washington, Mr. Truman got on both sides of the mid-century line with all the aplomb of a good politician. He told Congress early in January that "the state of the Union continues to be good," as it nears "the midpoint of the twentieth century." Later he spoke of moving forward into the second half of the century. He promised an infinitely better life for everyone, whichever side of the mid-century line, and reaffirmed his prescription toward that end: assistance for small businesses; more houses for the middle-income groups; tougher anti-monopoly laws, retention of rent controls; development of natural resources; better insurance against the hazards of old age, idleness and unemployment; better medical care and education, civil rights legislation; new labor relations law and production payments to support farm incomes. All this was to be coupled with suggestions on how to balance the budget.

The bloody and confusing 1940's had brought to most men's minds a fear of the future such as they had not known for centuries, and Mr. Truman's objective is to temper that fear. Only time will tell whether the terminal point of such a philosophy would or would not belie the old English proverb citing inability to both eat and have a cake.

Brave words to the contrary, the national budget persisted in sliding off into the red. The smell of further inflation was equally persistent. And just to impart a little more zip to the trend, the American Federation of Labor announced its intention of seeking "substantial" and "large" wage increases this year. The AFL forewarns of a business slump after June and a 10¢ an hr wage increase for each worker "will be enough to reverse the prospective downward trend and start a rise." Thus it goes—"the way to halt a drop in demand for labor is to raise the price of labor."

Elsewhere in the harassed world, beleaguered individualists in New York achieved a local triumph of sorts for one of the freedoms denied by the Soviet and Nazi systems—the right of escape, some escape at least, from the servitude to decibels. Grand Central Station weakened perceptibly at Christmas on its 5-min commercials and bore down heavily on "Hark, The Herald Angels Sing," and on New Year's Day capitulated entirely by abandoning the broadcasting of canned music and slugs of advertising slogans. In Europe, M. Bidault's Assembly majority fell away and newspaper accounts of the incredible monstrosities of political trials beyond the iron curtain were supplanted by headlines marking the birth of Princess Yasmin. The latter was surrounded by Byzantine splendor but otherwise



showed no apparent deviation from the accepted norm. In Western Germany, Dr. Adenauer began to speak up more boldly, and the whole fabric of Allied policy towards the defeated enemy further dossolved under the impact of rivalry with the Russians. In the Far East both the Americans and British seem somewhat embarrassed by Chiang Kai-shek's refusal to lie down. Dual thinking regarding the "squalid and useless struggle in the China seas" is such that newspapers give sympathetic accounts to an American ship running supplies to the Communists at Shanghai while calling for help for the recognized Chinese government in Formosa. And with Britain recognizing Communist China they may well have to team up with Russia against the United States to kick the Chinese representative off the UNO Security Council, the Far Eastern Commission and the Allied Council in Tokyo.

Far Eastern policy has never been a bipartisan matter and as General Chiang retreated to his final redoubts in Formosa and Hainan, a number of Republicans pecked away at Administration lassitude. Senator Smith advocated the occupation of Formosa and Messrs. Hoover and Taft asked for use of the Navy. Mr. Acheson has stood firm, however, against military aid. None the less, a study will be made of the Generalissimo's chances. Mr. Truman has a private purse of $75 million for the Far East and some $94 million in frozen ECA funds. Mr. Truman will likely trade some of this money for support of new Marshall plan appropriations, always mindful, of course, of Mr. Acheson's plan to drive a wedge between the Chinese and Moscow, and his wish to avoid any revival of imperialist nightmares throughout the Orient. Mr. Acheson is very convinced that the victories of Communism in China will be short-lived and South Asia will not fall

under the Soviet yoke if the West can win the friendship of the independent Asiatic people and convince them that real efforts are being made for their welfare and progress. It is tact and human understanding in official and unofficial contacts, and also Mr. Truman's Fourth Point, which will tip the balance and stem the communist flood—not the mailed fist.

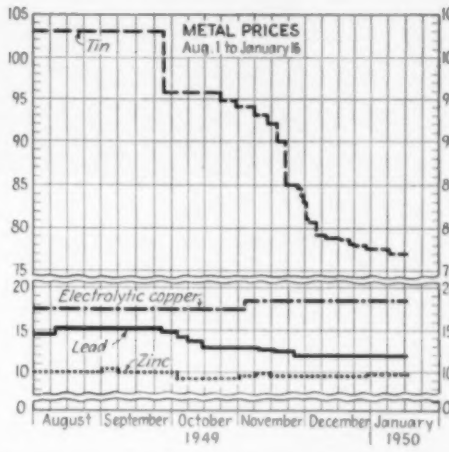
In Britain an electorate is getting readied for a general election in late February, the first since 1945. Despite the steady victories in local by-elections, the tide has tended to run steadily against the Labor Party. Ever since 1945 Dr. Gallup's British Institute of Public Opinion has periodically asked the question: "If there were a general election tomorrow, how would you vote?" The results are shown here in graph form. While Mr. Truman all too recently proved that election prediction is indeed an unproductive occupation, Dr. Gallup's figures have been accorded enough recognition to wring most of the optimism out of Labor politicians.

While England debates the merits of Socialism, the eyes of Western Europe continue to stray across the iron curtain with a certain degree of fascination. In Moscow the "genial father of humanity" celebrated his seventieth birthday amid an orgy of idolatrist vulgarity, mass obeisance and orations of flattery running the entire gamut of the richest superlatives of the Russian language, all interlarded with some genuine respect and sincere personal feeling. Only in Tito's Yugoslavia was the great day ignored—perhaps there all energy is being hoarded for a bang-up competitive celebration on May 25, Tito's birthday!

Stalin's birthday was marked by orations by Politburo members, all written by central headquarters and therefore indicative of the grand strategy of communist thinking. Apparently Stalin fully intends to prove anew the astonishing longevity of Caucasians, and furthermore, has no intention of retiring to an elder statesman role. He is and will be the master-mind behind unrelenting pressure on western countries. The revolution must not be permitted to lag. Flexibility and cunning are to mark all relations with the capitalist countries and attempts will be made to split the western bloc by dangling before the United States offers promising bans on atomic weapons and agreements to divvy up the world into spheres of influence.

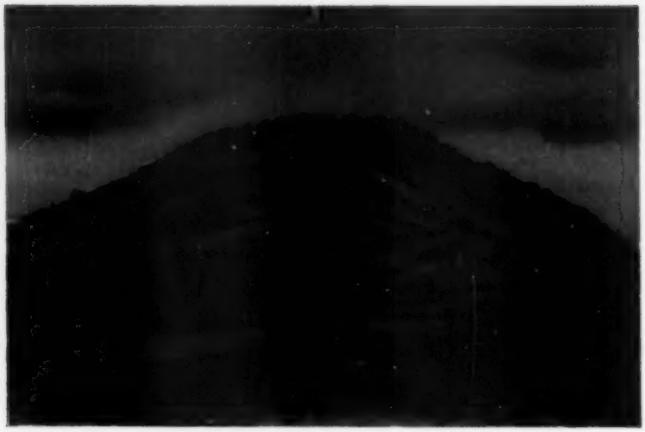
Down in Bulgaria the testimony released on the trial of communist Secretary-General Kostov indicates that the Soviet world is far less strong, far less monolithic than its champions claim. Apparently Kostov, a long-time dependable Comintern agent, refused at times to sell exclusively to Russia when better prices could be obtained for certain products in Europe. He also considered accepting Marshall aid and tended to view Tito's Yugoslavia as having "a sober and realistic policy." In satellite Russia such thinking leads straight to the hangman's noose. Kostov's failure to stick by his "confession" in the trial parody in the Military Club ballroom so infuriated and confused the functionaries as to make them speechless for 48 hr, until *Pravda* laid down the line by describing him as a "shrewd, experienced and determined enemy . . . soft and oily voice . . . cunning eyes of a thief." *Pravda* also derided Kostov's "bent back," not mentioning that the hunchback dates from 1924 when, as a communist organizer, he threw himself from the Sofia police building roof to escape torture. Elsewhere around the Russian periphery other nooses are being readied for other "diversionist" communist necks.

Back in the United States the stock market jumps around while industry as a whole operates steadily at its best level since last spring. Order books for most consumer goods are well filled; the steel industry is working at forced draft to plug up strike-reduced inventories, coal stocks need replenishing, and Detroit is scheduling higher activity for early 1950 than in the comparative 1949 period. Construction contracts remain very high. As the Treasury starts paying $2.8 billion of insurance premium refunds to veterans, there is every indication that industrial output during the winter will inch further upward.



As for base metal prices, the release of inflationary pressures is such that a general rise in nonferrous prices is not out of the question. In most base metals, excepting tin, American demand has recovered along with a firming in price tone. Consumption of copper has been maintained at a high level and deliveries from producers and custom smelters are heavy. The strength of the market is best gaged by hardening in the copper scrap market. Lead scrap and the forward price of zinc have also tended to rise. Tin, however, has persistently strayed to the downward side, with prices virtually dictated by the RFC which sells the output of the Texas smelter. Recently the British Ministry of Supply transferred 3,000 tons of tin to New York for the spot market, only to be met by the RFC announcement that it would be adjusted to compete with tin imports. So far the British government has shown no tendency to undercut the market and dispose of its stock. As other importers bring in supplies, however, the tin market is liable to show some of its prewar competitive dash and vigor.

Cerro Bolivar



Saga of an Iron Ore Crisis Averted

by T. W. Lippert
Manager of Publications, AIME

CUBA fancies herself the "pearl of the Antilles" and, by many, Jamaica is called "blessed." But far to the southward lies what is seemingly the Caribbean's most glittering jewel, the sparsely-settled, Spanish-speaking United States of Venezuela, indeed blessed by nature to a lavish degree. Not a large country, Venezuela is leaping from the donkey to the airplane under the impact of its natural wonders. Angel Falls, dwarfing Niagara and seldom seen by man, is hidden in the cold, primitive and unexplored Guiana highlands along the southeast border. A few hundred miles northwest are the low and torrid coastal plains from which gush unlimited black-gold streams of oil. About midway are the *Llanos*—low-lying, savanna-type grasslands, beneath which large deposits of bauxite are as yet barely touched by the explorer's diamond drill. And jutting starkly upward 2000 ft from a surrounding expanse of savanna is a small mountain (see front cover), 11 miles long and 1 mile wide. It is, or was, La Parida.

Picture above shows the topmost tip of Cerro Bolivar. The road (for diamond drills) was blasted out of the richest of solid iron ore. Shed is a pumping station.

This is the first disclosure that over half of La Parida is solid iron ore of almost unbelievable purity—by all sober estimate the richest and greatest iron deposit in the history of the world! The U. S. Steel Corp. has La Parida under denouncement for 100 yr, and its discovery was the terminal point of the most far-flung and greatest of all ore searches, and also the quietest. Both the discovery and the plans being rushed forward for its development constitute a classic example of free enterprise at its very finest—daring, imaginative and responsible. Within a few years this little mountain will be exerting a profound influence on the American steel industry and the entire American economy. Its existence certainly brightens the long-term military potential of the United States. The only possible jarring note in as dramatic a record of geological derring-do as anyone could wish for—hardly worth a quibble perhaps—is that La Parida (in prim translation "a new mother"), after being delivered of her first ton of iron ore for experimental tests, has found herself both re-sexed and re-named Cerro Bolivar. To the midwives, a solid, masculine hero's name is none too fitting for a mountain soon to rank with Mesabi in the lexicon of steel.

These last few years, amid all the turmoil of postwar strikes, investigations, and insatiable demands for production, the steel industry has occasionally twisted around to stare uneasily at the six great Superior ore ranges. Like Alice in Wonderland, these ranges seemed to shrink and grow with disconcerting rapidity, depending on whether the reserves were being estimated by Mr. Barloon of Western Reserve University or Mr. Randall of Inland Steel Co. From out of the blue have come fitful showers of statements regarding Taconite, for the most part statements barren of any real substance. There were reports of Bethlehem opening a new ore pit in Chile, of Bethlehem and the U. S. Steel Corp. in Venezuela,

and of Republic in Liberia. Up in Labrador, Hanna-Hollinger have mixed a succession of ore discoveries and dramatic press relations with exemplary skill. And down near Trenton, N. J., the U. S. Steel Corp. a few weeks ago announced acquisition of 3800 acres of farm land—quite possibly for a 2 million-ton steel mill within a few years, not in perhaps 10 yr as has been the general press assumption.

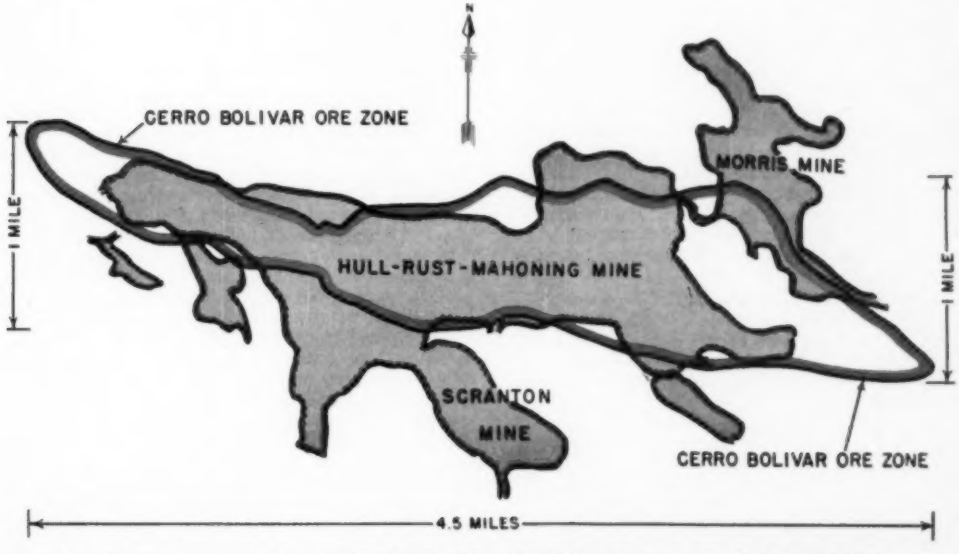
These individual events may seem only casually related, the usual cut and thrust of economic enterprise. But just like beads on a string, they all fit together. And by far the biggest bead on the string is the U. S. Steel Corp.'s Cerro Bolivar. Within four years shipments into Birmingham, Youngstown, Pittsburgh and Trenton could be coming out at the rate of 10 million tons annually, to be stepped up later to 15 million tons, all the richest type of blast-furnace grade. And hard openhearth charge ore, a really scarce grade already 800,000 tons a year in short supply in the Pittsburgh area alone, is scattered in very generous lenses over U. S. Steel's eighteen denouncements and six concessions.

This article tells for the first time the story of Cerro Bolivar, a story of men and events which sometimes reaches into the suburbs of improbability. This article also tells of the other beads on the string, all components of the future iron ore supply of the United States—Taconite, Labrador, the various ore deposits around the periphery of the Atlantic Ocean, and of course the historic ore ranges of the Lake Superior area.

Stretching across northern Minnesota, in an extended sheet, 100 miles long and several miles wide, lies a primitive iron formation, Taconite, of which the Lake Superior iron ranges were formed. There is not one Taconite but many types of complex mixtures of hematite, magnetite and waste. At best it is one-third iron and two-thirds waste. Sprinkled here and there in the Taconites, like rich raisins in a poor cake, are the high-grade ore pockets, the fabulously rich (at one time) Cuyuna, Mesabi and Vermillion ranges. The sheet of Taconite rises again in southern Ontario and a few more not-so-very-rich raisins stick out (Michipicoten, Gunflint, Atikokan ranges, *et al*) with Goulais the most recent finding in this area. (Algoma Steel Corp. just recently proved 50 million tons of magnetite minable by open-pit methods along the Goulais River.) Another short jump and there is a small rich outcrop at Steep Rock (Inland Steel is seeking additional deposits in this area), after which the Taconite falls away only to rise again along the far distant boundary line between Labrador and Quebec, which represents the last remaining large-sized explorable iron range in North America. Here the raisins pop out all over the place, relatively small pockets of rich ore outcropping over an area some 85 miles long and 15 miles wide. The best raisins are on the Hanna-Hollinger concession and their exploration has generated successive waves of publicity. This whole band of ore, from Fort Chimo on cold and desolate Ungava Bay down into Wisconsin, possesses iron formations of great similarity and was undoubtedly created in the same geologic age.

When Benjamin Franklin pressed down hard with his "strong red line" at the Treaty of Paris, it was Providence assisted by blind good fortune that set the ragged Canadian-American boundary line north of the Mesabi, Vermillion and four other fabulous iron ranges, which were discovered almost a hundred years later. Of late, with discovery of the Labrador pockets, historical justice at long last seems to be swinging a little toward a more equitable balance, but even so there is nothing in Labrador resembling the fabulous Hull-Rust pit at Hibbing, Minn.

Since the early 1900's over 80 pct of the nation's iron ore has come from the few pits in



Cerro Bolivar compared in area with the champion of Lake Superior open pits, Hull-Rust-Mahoning.

the Lake Superior district. The country has been built and two world wars fought and won on the rich, easily-mined ore from these ranges, with the Hull-Rust pit carrying a large portion of the load. Since the ranges were first worked there has been available "unbounded" and "unlimited" iron ore, rich and for the most part open-pit. It is somewhat ironical to note that back in 1913 the U. S. Steel Corp. considered itself so burdened with iron ore that it hastened to rid itself of the "Hill Lease" containing well over 300 million tons. In the mid-1930's many steel companies chafed under the tax loads levied against their ore properties and foresaw no ore problem for their children or even their children's children. Even as late as 1939 many steel companies viewed ore reserves primarily as an *embarras de richesses*, but the U. S. Steel Corp. began shaking itself loose from the universal somnambulism and announced its willingness to sell ore in the open market. This historic move is a good case study of responsibility —it was a basic decision to share greater reserves with the remainder of the industry. Some 52 million tons of ore has been sold to all comers. If depletion and crises were to be the end result, at least everyone in the industry would approach that grim accounting at about the same time.

The second world war chopped 340 million tons of the highest quality reserves out of the pits, and a couple of years ago C. M. White, president of Republic, gloomily announced that no more wars were ever going to be fought off Superior's natural ores. He might just as gloomily have announced that there weren't many more years of peacetime prosperity left in the Superior pits. Nature's lush legacy had been spent, and many a directors' room echoed to grim forebodings, although on the surface the attitude was one of calm confidence. The safety of the country, the prosperity of the country and the preeminence of the American steel industry were approaching the cross-roads, and the disinclination to engage in barren controversy and discussion in public was based not on a Micawber-like attitude that "something will turn up," but genuine confusion as how to develop the technology and raise the great sums of money necessary to fend off the crisis.

The usable iron ores in the Lake Superior district, at present rate of consumption, will be totally exhausted in about 1963 for Independent companies and in about 1970 for the U. S. Steel Corp. And the assumption in some quarters that there are some hidden reserves being held back for tax reasons is a piece of dangerous wishful thinking.

As of Jan. 1, 1950, the natural iron ore reserves of the Lake Superior district amounted to 1290 million tons. Of this quantity the U. S. Steel Corp. held 690 million tons (53.5 pct). Among the Independents, National Steel is by far the best off, whereas one major steel company has less than 20 million tons in reserve and several others have less than 40 million tons each. An additional discouraging factor is that open-pit reserves (the easiest to skim off during a war) have fallen to 70 pct of the total reserves, and the industry must of necessity face up to more and more costly burrowing underground to get at their ore.

Some thirteen to fifteen years yet to prepare for total exhaustion might seem to many an outsider to present perhaps a rather reasonable breathing spell in which to somehow lick the problem. The task, however, is definitely not one for the next decade. It has demanded immediate solution. As a military compulsion alone some of the free-shipping ore remaining in several Superior pits should perhaps be frozen as a national emergency stockpile. With little over 1 billion tons up there, and conscious of the fact that nearly ½ billion tons were required in the last war, the situation becomes one that might adversely affect the life and future of each citizen. Further, if some optional source of metallics is to be devised, the tremendous capital investment required had better be scaled over as long an interval of time as possible, not come in one fell swoop. Even at best there will have to be some finger scratching along the bottom of the money barrels to carry the investments in prospect. Thirdly, and very important, indeed, steel companies must know where their metallics are coming from 15 yr hence in order to determine what equipment to build now and where to locate it. Costly ore boats cannot be constructed now only to have no cargo to haul before they have outlived their economic life. The same applies to blast furnaces, openhearth shops and rolling mills, which, if mislocated, could easily sink a steel company financially when the going got tacky.

The only two ore options available have been first, to devise some technically and economically practicable method of exploiting the huge Taconite deposits of the Superior region and, second, to go to foreign sources for ore. And, quite obviously, the Taconite challenge (Comparable in scope, complexity and cost with the war-time synthetic rubber program.) must be met, for a great nation like the United States with the world's largest steel industry can hardly risk drifting into a position of complete dependence on ore sources in foreign countries.

There's iron from which to make steel in the Taconites of Lake Superior—plenty of it, many billions of tons. This truism has led to a variety of public statements that the country has plenty of ore and that any concern is unwarranted. But the winning of usable iron from Taconite involves technical problems of such complexity that until very recently the huge sums of money required in capital investment and operation made a very large increase in the selling price of steel seem inevitable. It would be questionable whether the country's economy could stand such an added load, for industrial health over the past half century has been based in great part on the availability of large quantities of cheap steel.

Taconite is a mixture of hematite and magnetite crystals held in a matrix of rock which may be original or partly (or completely) decomposed. In most instances the waste accounts for about two-thirds of the total bulk, and since all the iron is usually not recoverable, it is necessary to mine close to 4 tons of material to get 1 ton of iron. There can be proportionally a little hematite and a lot of magnetite, or vice versa. The crystal size of either iron mineral can vary in any variety of combinations, and the proportion of ore to waste material can vary from infinitesimal to a rather respectable figure.

All pilot plants to date handling Taconite approach the problem in pretty much the same way. The Taconite is ground to a fineness dictated by the crystal size of the magnetite, and the magnetite is separated from the powder by passing over a drum or belt-type magnetic separator. The hematite goes along with the waste in the tailing. Then this powder must be reconstituted or agglomerated in some manner—nodulizing, sintering, pelletizing—to get a lump suitable for blast furnace charging containing about 63 pct iron. Obviously the most desirable type of Taconite should have, say, two-thirds of its iron in the form of magnetite and be characterized by relatively coarse crystals which would require grinding to much less fineness and make magnetic separation easier. Whether the Taconite has to be ground to 100 mesh or to 300 mesh, or whether the magnetite content is one-third the total iron or two-thirds, are the variables determining whether costs are to verge on the prohibitive or be modest enough to avoid serious increase in finished steel prices.

Fortunately, the eastern end of the Mesabi range has large deposits of Taconite which can be handled by the relatively cheap magnetite concentration process. The ore is hard and expensive to drill, but Minnesota has reduced taxes on this Taconite in order to encourage such operations. Michigan's Taconite, unfortunately, has hematite as the predominant iron-bearing mineral. Of course, this nonmagnetizable hematite can be reduced to magnetizable magnetite by heating and using reducing agents such as carbon monoxide. So far an operation of this complexity involves very discouraging cost factors.

Up on the ranges it is said that Taconite will be "manufactured" rather than mined. It is self-evident that the time, money, men and equipment necessary to drill and blast ore, grind it to a powder, extract one-quarter of the powder as a usable product, and then reconstitute this quarter into a lump form, are all on a far higher order than the conventional practice of having a steam shovel lift out great bites of hematite for direct shipment to blast furnaces. Furthermore, the problems involved in rapidly expanding Taconite concentrates during times of wartime national emergency are rather formidable, and may demand for minimum safety a number of standby plants or strategic stockpiles of concentrates.

At the very least, investment cost for Taconite concentrates will be well above $20 million per million tons of annual production of lump concentrates. This is investment cost alone which must be amortized and does not take into account day-to-day operating costs. If Taconites alone were to be depended upon to supplant the dwindling natural ores of the entire steel industry, some 52 million tons of capacity would be required in the next 15 yr. This would necessitate an investment of over $1.4 billion, a staggering sum which in final analysis would necessarily be borne in great part by steel consumers.

Even though the whole burden of future ore supply will not be thrown onto Taconite, a number of companies have attacked the problem with a zeal that mounts in direct proportion to the depletion of their natural ores. Within a few years the whole pace of Taconite exploitation must of necessity step up to a much faster tempo.

The Erie Mining Co. (Bethlehem, Youngstown and Pickands, Mather & Co.) began Taconite research some 10 yr ago and now is in the second year of operation of a 200,000-ton annual pilot plant at Aurora. From all information that can be gathered, the agglomeration of fine concentrates is presenting difficulties and there has been little or no production to date. Even so, this group of three companies now has plans in progress for a new plant of 2½ billion-ton annual capacity of lump concentrates.

Another combination, Reserve Mining Co. (Armco, Wheeling, Republic, and Cleveland-Cliffs) has acquired a 1½ billion-ton Taconite deposit, averaging 24 pct iron, at Babbitt, Minn. During the past 3 yr it has completed plans for a 10 million-ton capacity plant. Application was made for a $50 million loan from RFC to cover initial steps of constructing a 2½ million annual ton plant. The requested 30-yr loan was denied, and the RFC has suggested a 10-yr loan. This is a good example of the financing problems involved in shifting to Taconite—as yet RFC and the government is not conscious of the full seriousness of the situation.

Jones & Laughlin has completed a research laboratory for Taconite study at Ishpeming, Mich., and has acquired deposits of ore in Michigan and Ontario, Canada.

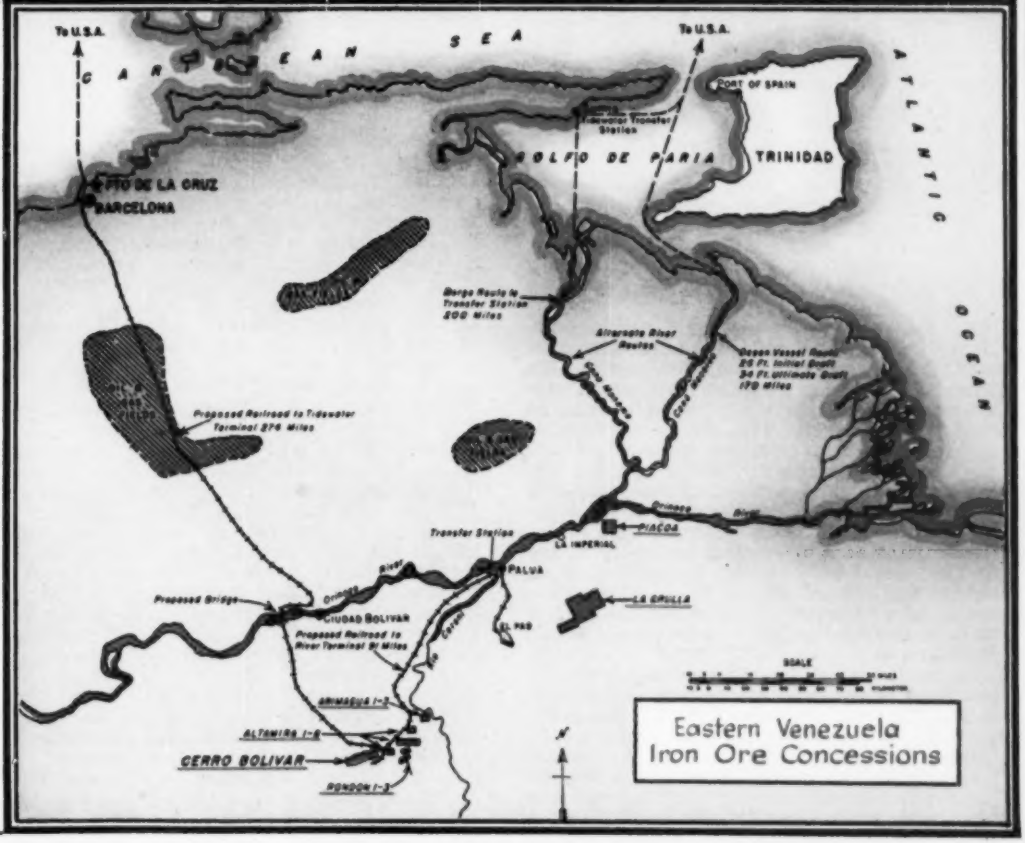
Cleveland-Cliffs has also completed a research laboratory at Ishpeming, Mich. This company plans to build a Taconite pilot plant in Michigan next year.

The U. S. Steel Corp. has Taconite reserves to produce some 300 million tons of concentrates in the western Gogebic Range in Wisconsin. However, much of the initial Taconite production will be based on the Minnesota ranges, and it is conservatively estimated that the district Taconite holdings will produce 1¼ billion tons of shipping lump concentrate, averaging 63 pct iron. The U. S. Steel Corp. definitely plans to have a pilot plant in operation by late 1952 turning out 500,000 tons of concentrates annually. This corporation has a very large laboratory at Duluth operating on Taconite research, and in late months officials are of the growing opinion that they have their problem pretty well licked. This is hopeful news, although it should be noted that large proportions of U. S. Steel's Taconite holdings are of the ideal type—large crystals and a very high proportion of magnetite.

In the next 20 yr it is to be expected that independent steel and ore companies must of necessity get into production some 17 million tons of Taconite concentrates. In the same period of time the U. S. Steel Corp. might well develop about 10 million tons of beneficiated Taconites, the exact capacity being dictated by cost relationships with Venezuelan ore imports delivered to various plants. Such an industry program, which cannot be avoided, will require new capital investment of well over $½ billion, and to manage such a sum will require some good profit years as well as a long series of special financings. It is certain that the growing dependence on Taconite will be reflected somewhat in finished steel prices. Until recently this figure looked as if it may be well over $4 a ton, although of late there is growing hope that when big production plants get into operation the

Water & Land Transportation Routes From Venezuela, Labrador, Quebec & Lake Superior District To U. S. A. Furnaces

✻ Location of U. S. Steel's projected eastern seaboard steel mill, to more economically service part of the 7 million ton annual finished steel market in that area. Assuming an annual 2 million ingot ton plant (3 blast furnaces; 16 openhearths; continuous hot and cold mills for plate, strip, sheet and tinplate; mills for wire, pipe and bars), the plant's completion would obviously coincide (before 1955) with availability of adequate Venezuelan ore. Giant ocean ore carriers would come directly to the plant (see map), barge trans-shipments would arrive from Baltimore via the C. & D. canal (map, dotted line). Quite possibly another "South Chicago" could rise on these flat New Jersey farmlands south of Trenton.



higher cost factors that must reflect down to the finished steel may be considerably less. In any case, Lake Superior Taconite concentrates will show up best competitively, vis a vis foreign ores, in the Lake Michigan and Erie consuming centers.

While the sharp-pencil experts in the back rooms worry the technology and economics of Taconite, a number of steel companies are reaching out into foreign countries to locate and acquire rich direct-shipping ores. While the problem for the nation as a whole is that of getting usable iron in any form and at any cost, the individual steel company has both that problem and also the one of competitive survival. And the individual plant's competitive survival will be based on cost of hot metal out of the blast furnace, vis a vis his competitors' costs in any particular district. In some instances the blast-furnace charge may be almost all Taconite concentrates, as, for instance, those plants bordering the Great Lakes to the west, such as Chicago and Detroit. Other plants may base their operations for the most part on foreign ores, such as plants in Birmingham and other Gulf areas, along the Eastern Seaboard up into the Northeast sections of the country. Still others, the great producing plants in the Pittsburgh, Youngstown, Cleveland, Wheeling area will likely fall back onto various combinations of Taconite concentrates and foreign ores.

It is probable that the rich Venezuelan ore discovered by the U. S. Steel Corp. will be available in almost any producing center at a better competitive price than units of iron from any other source. This great lode of ore which can practically be pushed onto railroad cars will be available to all who wish to buy. Further, this one body alone could supply the entire American steel industry all through the lifetime of children now being born. However, for reasons already mentioned—military, competitive, etc.—development of Cerro Bolivar will likely never rise beyond 15 million tons annually, although its price laid down in American steel centers will be a yardstick exerting great influence on iron from other sources.

For many years some iron ore has entered the United States from foreign countries—Bethlehem's large shipments from Chile and smaller deliveries from Cuba are the most conspicuous example. During the 1930's foreign ores and foreign finished steel were readily available at low prices, and some filtered throughout the eastern areas of the country. But they never got into the great midwest industrial areas primarily through inability to enter the Great Lakes without transhipment. Certain industrialists in the Great Lakes area based their opposition to the St. Lawrence Seaway on their desire to minimize foreign competition. Of late this opposition has ceased as a reflection of the imperative need to bring in foreign ores, particularly Labrador ores, as cheaply as possible.

Since Bethlehem has always been the most persistent of ore importers to feed its Sparrows Point, Md., plant, it might be of interest to review the foreign holdings of that company.

From Chile, Bethlehem imported 2.6 million tons of iron ore in 1948. This company recently has commenced expansion of its El Romeral iron mine as the result of a $2,750,000 credit from the Export-Import Bank for the Chilean Development Corp. Estimated reserves in this area run to some 20 million tons of ore of 60 pct iron.

Bethlehem has for years also been in Cuba, bringing out small quantities of lateritic iron containing nickel and chromium. In general, Cuban laterites are not competitive with other ores, and persistent surveys by several companies have never uncovered any high-grade iron deposits of sufficient size to warrant interest as a dependable source of supply. Small deposits occur along the west coast of Pinar del Rio Province containing possibly 1 million tons of open-pit ore averaging 55 pct dry iron and 10 pct silica.

The large deposits of lateritic iron ore containing nickel and chrome are in the Moa and San Felipe districts. U. S. Steel has some long-time holdings in this area, and there are other deposits which never have been denounced. Until an economic method is devised for extracting the nickel and chromium, these lateritic ores will not assume any importance. It might be of interest to note that Nicaro Nickel Co., financed during the war by the U. S. Government and directed by Freeport Sulphur Co., used these laterites from which to win nickel. This operation is now discontinued. For a few years some 3600 tons of ore carrying 30 pct iron was used daily. The iron was reduced to magnetite chemically and was wasted along with the chromium and tailing. There is quite a tonnage of iron lying down there with the tailing, and if some economical method could be devised to remove the chromium it would be possible to recover, through magnetic processes, a suitable ore for blast furnace use by agglomeration.

Bethlehem also held by far the best deposit of iron ore on the West Coast of Mexico, at Las Truchas. There is probably a large amount of open-pit ore in that holding, and Bethlehem first acquired possession by virtue of denouncements made prior to 1926. Bethlehem subsequently lost control of the deposit but re-acquired it under a special concession contract with the Mexican Government. It is believed that the contract has within the past several years been brought up for revision and perhaps has reverted to the original owners. Las Truchas has never been exploited and may never be until some company uses it for a hot-metal operation along the West Coast of the U. S.

Quite possibly Bethlehem's most interesting foreign holding is in Venezuela, where for the past ten years it has had a concession staked out at El Pao (see map). M. A. Hanna had been in and out of Venezuela for years and at one time had El Pao under option prior to Bethlehem's acquisition of the property. Hanna still has concessions next to El Pao. It would seem that M. A. Hanna Co. was singularly unimpressed with its Venezuela holdings even though their deposits, called Murcielago and Carambola, offered promise of possibly 10-20 million tons of open-pit ore. Bethlehem appeared satisfied with its El Pao holdings and showed little interest in the Represalia deposit, about 100 miles away and containing less than 5 million tons of ore, or other ore occurrences along a belt 100 miles long and situated between the Represalia and El Pao deposits.

Bethlehem's El Pao open-pit mine in Venezuela has reserves of at least 60 million tons, and

with an expenditure of some $50 million plus, this company hopes to be shipping out some 2.4 million tons annually by possibly late 1951. A lot of this ore is of the hard, crystalline type, open-hearth grade rather than blast furnace grade.

By late 1946 Bethlehem had practically completed its loading dock facilities at Palua near San Felix, at the confluence of the Orinoco and Caroni Rivers, and was building the railroad from Palua to the El Pao mine. By April of this year (1950) a transfer docksite will be finished and ready for operation at Puerto Hierro, east of Guiria on the north shore of the Gulf of Paria not far from Trinidad. They plan to get the ore out by using six barges each of 4000-ton capacity (300 x 50 x 6 ft) and a draft of 14 ft when full loaded. Two ocean-going tugs have been purchased and these tugs will haul two barges at a time from Palua down the Orinoco and the Cano Manamo past Pedernales (see map) and across the Golfo de Paria to the Puerto Hierro unloading station. There the ore will be loaded onto 26,000-ton ocean carriers—the specially designed Venore type. Bethlehem is also building several ships of 6000-ton capacity with a fully-loaded draft of 17½ ft for use on the same haul as the barges. Apparently the two methods of transportation are to be compared.

This shipping operation devised by Bethlehem is the most ingenious possible for the amount of ore they plan to move. The entry of U. S. Steel Corp. into the same area with a very much larger operation might, of course, influence future shipping plans. Certainly Bethlehem will finish its railroad to Palua near the confluence of the Orinoco, a distance of some 40 miles. However, if U. S. Steel or the government chooses to dredge the Orinoco for the largest of ore carriers, it would seem that Bethlehem also would bring its 25,000-ton Venore carriers right up to the railroad dock. However, if the Corporation elects to avoid river transportation and drive a railroad 274 miles across country to a tidewater terminal at Barcelona, then Bethlehem could possibly make some arrangement to use the same railroad or continue with its planned barge and transfer arrangement at Hierro. The Corporation's decision as regards railroad vs. river dredging probably will be reached before long.

Before describing in detail the U. S. Steel Corp.'s search for foreign ore and discovery of Cerro Bolivar, it might be best to cover two other foreign holdings—Liberia and Labrador.

African ores are becoming an increasingly important factor in the world markets. In Liberia various explorers were looking over ore showings there as long ago as 1938. There are occurrences of high-grade ore but the tonnages are in general too small to create a great deal of interest. Geologists from the U. S. Geological Survey examined the same area in 1944. Republic is now definitely planning to bring ore out of that country, probably for use in the Alabama area until the St. Lawrence Seaway permits shipment directly into Cleveland. Port facilities for loading ore are now being constructed at Monrovia, the capital city of Liberia. Republic has acquired majority interest in the Liberian Mining Co., controlling some 20 million tons of ore, and it is believed that the company plans to ship 500,000 tons annually, beginning sometime in 1951. The development cost will be on the order of $8 million, of which one half the amount will be an Export-Import Bank credit.

The Labrador iron ore deposits have attracted a great deal of publicity during the past few years. As already pointed out, there is considerable ore of high quality up there, although as yet no definite plans are under way to get it out. There are some rather formidable obstacles facing open-pit mining operations and delivery of the ore over about 400-500 miles of tundra to a St. Lawrence River port. The railroad alone will cost more than $100 million, and at the mine the amortization of equipment must of necessity take into account use for only about half of each year. Personnel at the mine will also be somewhat of a problem, with the two possibilities of their being idle half the year or some provision made to bring them out for the non-mining period.

Without the St. Lawrence Seaway, the Labrador ore will likely not be competitive at most consuming centers with either other foreign ores or Superior Taconite concentrates. With the St. Lawrence Seaway, this ore should have a comparative cost factor through the Cleveland area and down into Youngstown.

By late 1946 it was already apparent that an important iron district would eventually be developed along the boundary line between Labrador and Quebec, primarily on concessions controlled by M. A. Hanna Co. and the Hollinger Consolidated Mines. The Labrador territory is part of Newfoundland, which is governed by a commission deriving its authority from the British crown. The Quebec area is under the Dominion of Canada.

This iron formation is part of a rock series occupying a trough which extends for about 350 air miles northward of the St. Lawrence River. Its northern extremity at an elevation of 125 ft is only about 60 miles from Fort Chimo, a tidewater port on the Koksoak River which enters Ungava Bay. Practically all known iron formations in this belt (see map) are now covered by exploration licenses or concessions, as follows, from north to south:

In Quebec

(1) Quebec-Labrador Development Co., Ltd.

(2) Fort Chimo Mines, Ltd. (Frobisher).

(3) Norancon Exploration (Quebec) Ltd. (Noranda, Anglo-Huronian & Conwest).

(4) Hollinger North Shore Exploration Co., Ltd. (Hollinger & Hanna).

In Labrador

(5) Labrador Mining & Exploration Co., Ltd. (Hollinger & Hanna).

(6) Norancon Exploration, Ltd.

(7) To the East of this iron formation belt Norancon has two concession areas in Labrador.

(8) Dome Exploration (Canada), Ltd., also has a concession area north of the Hamilton River bordering the Atlantic Ocean. No iron formation is expected to be found in this latter area.

(9) Kennco Exploration, Ltd. (Kennecott Copper Co. & N. J. Zinc Co.). A titaniferous magnetite deposit much further south (see

map), about 50 miles north of Mingan on the north shore of the Gulf of St. Lawrence.

Kennco has disclosed the presence of some 50 million tons of reserves containing 35 pct titanium. To get the titanium this company has announced plans to spend $25 million to develop the deposit and another $15 million for an electric smelter of initial capacity of 1500 tons per day to be constructed at Sorel, Quebec, scheduled for operation by 1951. The by-product pig iron production might well range between 100-200,000 tons annually, analyzing 96.5-97 pct iron, 2 pct min C, 0.1 Si, 0.03 P, 0.10 or less S, 0.04 Mn, 0.2 Ti, 0.05 V and 0.05 Cr.

The Hanna-Hollinger concessions (Nos. 4 and 5 above) are the only ones on which important

explorations have been carried on. Some of the ore has been described as being physically like the harder ore of the Iron River district of Michigan, while other ore in the area has been compared with the Marquette range soft ore. In any event the ore will carry appreciable amounts of free moisture. In early 1946 there were reports that drill holes to a depth of 200 ft were still in frozen ground, thus indicating a nasty problem of handling frozen ore which would be gradually thawing en route to the furnaces. However, by early 1947 this possible difficulty was laid to rest when one exploratory tunnel, still in ore, penetrated 85 ft of ore under an ore cover of 70 ft without encountering any frozen ground except at the portal.

By early 1947 the Hanna-Hollinger geologists had discovered more than 25 ore bodies scattered over a length of 75 miles, with some of them ranging 1000 ft in width. By this time estimates were being made of a possible reserve tonnage sufficient to yield an annual production of 10 million tons for many years. Much of the ore showed an average 60 pct combined iron and manganese, with some showings containing up to 20 pct manganese. The group was granted a charter for the proposed Quebec North Shore & Labrador Railway Co., capitalized at $2 million. Financing of the remaining $100 million or so is as yet unresolved.

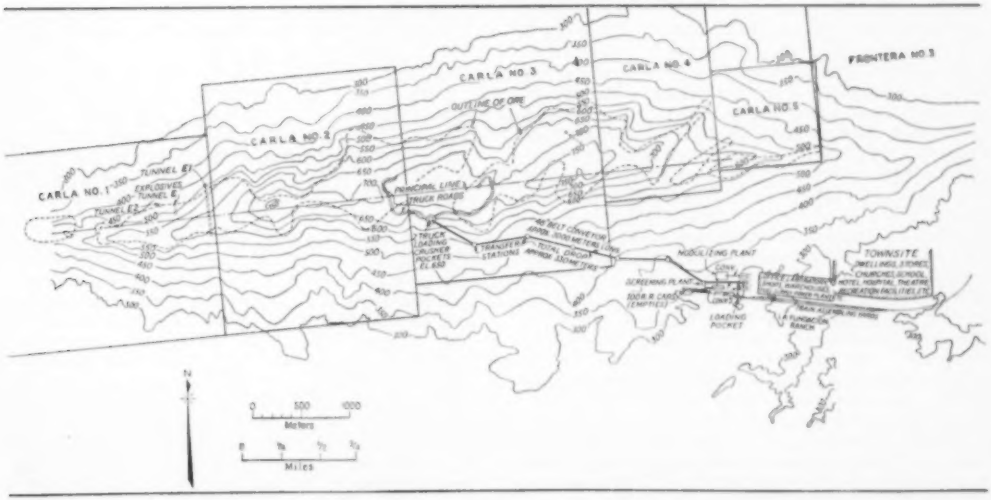
By late 1947 Hanna-Hollinger had discovered over 40 ore bodies and actual diamond drilling had proved tonnages in excess of 100 million tons. At this time it was estimated that further drilling would prove 300 million tons. Toward the end of 1948 over 44 separate ore bodies had been discovered with reserves of over 300 million tons indicated. In general the individual ore bodies are rather small and scattered over an area of 85 by 15 miles. The glacial-drift overburden is of the order of 16 ft.

A number of steel companies (Republic, Armco, National Steel, Wheeling and Youngstown) have joined with Hanna-Hollinger for several more years of exploratory work, and Republic has a one-year option for a quarter interest in this development. In view of the problems involved in building the long railroad to the St. Lawrence over very difficult terrain and delays certain to bedevil the St. Lawrence Seaway, it is likely that no large tonnages of Labrador ore will come out until the late 1950's. However, by 1960 these transportation problems will certainly have been rationalized and ore should be coming out at the rate of 10 million tons annually. To finance such a program will require a capital investment well over $175 million, exclusive of the lake vessels required to transport the ores from proposed railroad terminal at Seven Islands, Quebec, to the steel-producing areas.

The only other Labrador-Quebec concession given much exploratory attention has been the Fort Chimo area owned by Frobisher Exploration Co. It covers some 1000 sq miles and contains a length of about 35 miles of explorable iron formation located 100 miles north of the Hollinger holdings. It lies 100 miles southeast of Fort Chimo on Ungava Bay tidewater, the rail distance being about 150 miles with a drop of only 200 ft to this possible shipping port which is ice-free from four to five months of the year and lies 3000 miles from Baltimore. Late in 1946 the U. S. Steel Corp. was showing some interest in this concession, but a year later interest understandably waned as the Venezuela strike appeared to be of mother-lode proportions.

U. S. Steel's spectacular good fortune in Venezuela is the terminal point of a whole chain of encouraging and discouraging events leading back to Pittsburgh early in 1945. At that time, J. G. Munson, vice-president in charge of raw materials of the Delaware Corporation, along with B. H. Lawrence, then chief engineer, and R. E. Zimmerman, vice-president in charge of

Plan for initial development of Cerro Bolivar. The belt conveyor will generate enough electrical power to supply much of the area requirements.



research, became convinced that the Corporation would soon have to earmark extremely large sums of money for Taconite plants if the future ore supply were to be assured. It occurred particularly to Mr. Munson that at least one more look should be taken at all the possible foreign ore sources around the entire periphery of the North and South Atlantic Oceans. This was a momentous decision, particularly so since the Corporation's files were crammed with tons of reports on the known deposits, some dating back to 1908. Even though the projected search would be the most detailed ever carried out and involve the expenditure of many millions of dollars, board-chairman Irving Olds and president B. F. Fairless likewise okayed the gamble, although the odds were very much that nothing new of interest would develop.

By early 1946 field forces were in Brazil, Venezuela, Honduras, Guatemala, Puerto Rico, Cuba, Mexico and Sweden. Hundreds of men were beating far back in the hinterlands following up every known showing of ore and tracing down every rumor of red earth. Before the entire search was concluded thousands of men became involved, many hundreds of ore bodies had been drilled and explored in the Agalteca deposit in Honduras; in various parts of Guatemala; at Mayagues and Jancos in Puerto Rico; various showings in Honduras; all around the vicinity of Tunki in Nicaragua; all the possibilities along the east coast of Mexico, and more than 40 deposits along the west coast; the well-known ore fields in Sweden; the Wabana Mines in Newfoundland; Little Whitefish Lake in Quebec; Texado Island, Iron Hill and Zeballos in British Columbia; the Marampa and Tonkolili deposits in Sierra Leone, West Africa; holdings owned by Cia. Espanola de Minas del Rif in Morocco; the Djebel Djerissa holdings in Tunisia; the Ouenza ores in Algeria; the Conakry ores in French Guiana; several ore possibilities in Alaska. The objective was to find an ore body, preferably in the Caribbean area, of sufficient quantity and quality to justify full-scale development over a period of many years. The cost of ocean freight would represent a large part of the delivered cost of any foreign ore because of the large investment in vessels of special design and the high cost of operation of merchant ships.

The rich ore deposits of Brazil were given another looking-over, hardly necessary inasmuch as the Corporation had periodically surveyed these ores ever since 1930. None the less careful new investigations were made of the Itabira district; Candu, Saraiva and Nhotin, near Bello Horizonte; Casa de Pedra, near Congonlias de Compa; Santa Maria, near Macapa on the north bank of the Amazon; Cera in the northeastern section; and Amapa, near the mouth of the Amazon. However, all this searching around was not in vain. An excellent manganese deposit was uncovered in the Urucum Mountain, 15 miles south of the city of Corumba in the State of Mato Grosso. Reserves of 27 million tons are indicated, of 46.4 pct Mn and 10.7 Fe. U. S. Steel plans to pull out 100,000 tons per year. There were some interesting findings in various countries, but, from the very first, the Orinoco iron ore district in Venezuela seemed to offer interesting possibilities for the discovery of commercial iron ore deposits reasonably accessible to river transportation and of sufficient size to justify utilization. Late in 1945 air observers started to work over the area adjacent to and south of the Orinoco River extending from Ciudad Bolivar towards the Atlantic Ocean, a distance of about 160 miles (see map).

A year later, toward the end of 1946, some ore holdings around Bethlehem's El Pao concession and the M. A. Hanna concession had been looked over by magnetic survey with nothing of consequence showing up. Field reports and the aerial observation had concentrated attention on Piacoa and an area northeast of Upata. It was soon determined that the Upata area had various high-grade bodies averaging 65 pct iron, but they were narrow and not sufficiently extensive to justify exploration for open-pit mining.

The Piacoa ore is located about 100 miles downstream on the Orinoco from Ciudad Bolivar and approximately 75 miles from the mouth of the river. Piacoa is an insular hill which is 2 miles away from the Orinoco River. It is 1½ miles long and rises 550 ft above the surrounding flat topography. Ore occurs in a ferruginous quartzite with a grade from 40-50 pct iron. The ore would have to be concentrated, although the concentration problems would not be difficult, requiring only grinding to 28-48 mesh to produce a concentrate of 60 pct iron or better. Oil is within 50 miles and gas 150 miles. Diamond drilling got well under way early in 1947. Quite obviously Piacoa was nothing to get really excited about.

It is to be noted from the map that all the area being worked over was south of the Orinoco, east of the Caroni (a tributary of the Orinoco) and toward the Atlantic Ocean. The Venezuelan government had all this territory listed as a Federal Reserve Zone on which concessions of only 40 yr were possible, and even they had a number of restrictions. Venezuela considered this area the only possible iron ore area, and for that matter so did Bethlehem and M. A. Hanna, both of whom had been over the ground pretty thoroughly. Some of the U. S. Steel Corp. men were discouraged, some resigned and the whole Venezuelan exploration began to sag.

At this point Mack C. Lake, a long-time M. A. Hanna geologist, was borrowed by the Corporation to take charge of the field work. Mr. Lake had been half-retired and was living in California when M. A. Hanna called him back to do consulting work on their new Labrador properties. He was loaned to the Corporation to do work for them only in Venezuela inasmuch as Hanna was rather convinced there was nothing more of consequence there. Mr. Lake and two young geologists on the field team, K. Burrell and F. Kilstead, conceived the idea of aerially photographing the area south of the Orinoco and *west* of the Caroni, an area open to denouncement, not particularly remote and generally considered of no promise by both the Venezuelan government and the other steel companies that had previously looked over the territory.

Christmas Day, 1946, Messrs. Burrell and Kilstead were in Pittsburgh giving S. G. Munson a sales talk on spending another $185,000. Mr. Munson took another momentous gamble, gave his permission, and shortly thereafter Fairchild

was hired to take the photographs and Venezuela had granted permission to bring them out of the country for piecing together and examination.

Examination of the photographs got the geologists interested in two small mountains or hills, known locally as La Parida and Arimagua. La Parida was located about 50 miles due south of Ciudad Bolivar and rises about 2000 ft above the surrounding savanna topography, and is about 11 miles long. Arimagua is about 30 miles east of La Parida, nearer the Caroni River. Neither mountain is particularly remote and they both appear on many an old map. Early in the year a field crew started in to look them both over, with S. G. Sargis (a magnetometer expert brought in from Geneva Steel) in charge of the geophysical work.

In April of 1947 the field party found huge outcrops of iron ore all over La Parida hill and slides exposing large expanses of iron ore at Arimagua. As Mr. Sargis worked over the deposits with his magnetometer he recorded extremely high readings, the highest responses ever recorded in the world! The excitement mounted. There was no doubt that a really enormous tonnage of iron ore was there and it appeared of sedimentary origin, and of a grade in excess of 60 pct iron.

When the word got back to the headquarters town of Ciudad Bolivar, Mr. Lake decided to pull in diamond drills from about everywhere. The important thing was to determine as quickly as possible whether the high surface grade extended down to any reasonable depth or whether the magnetometers were recording an enormous lump of poorer ore and Taconite. The two Longyear diamond drills at Piacoa were started on their way to La Parida, two others on their way from the United States were diverted and another drill in Honduras was called for. It was planned to get at least five drills working at La Parida.

It was also imperative to survey the area well by magnetometer to lay out the boundaries of denouncements. And quietly, too. For the area was outside the Federal Reserve Zone and open to anyone who filed first. By this time some of the news must have seeped out for a couple of local entrepreneurs started to look around in the same area. Everything went well, however, and the Corporation quietly filed its Carla, Arimagua and Rondon (other ore showings in same general area) denouncements, which were published in the Government Gazette. On April 11, 1947 Carla Nos. 1 to 5, totaling 11,330 acres were accepted by the government. On Sept. 1, Arimagua 1 to 3, totaling 2960 acres, were accepted. Rondon 1 to 3, totaling 3700 acres were not accepted until early in 1948.

Drilling at La Parida actually got underway in July 1947, with ten American drillers on the job working for the contractor, E. J. Longyear Co., and headed in the field by Mr. Axel Auguston. Mrs. Auguston (Betty) officiated as cook.

By late 1947 reports coming from the field indicated iron ore outcrops on La Parida every few hundred feet for a distance of at least 3 miles. It was exposed in slides for a height of 200 ft. It still seemed to be of sedimentary origin with some secondary enrichment of low-grade horizons. Evidence indicated that certain high-grade horizons had been hydrated near the surface, hence lowering the grade. The hydrated material appeared to contain nearly the theoretical maximum of iron. Six holes totaling 1545 ft had been drilled, indicating a dry analysis of 63.4 pct iron, phos content decreasing from 0.175 pct at surface to 0.048 pct at 380 ft. Likewise, the loss on ignition decreased from 12.1 pct to 1.2 pct at 380 ft. The general conclusion was that this ore was similar in type and mode of occurrence to the thin-bedded hard ores of the Minas Gerais region in Brazil.

By early 1948 Mr. Sargis had run magnetometer readings over Bethlehem's El Pao deposit to check his readings with known underground reserves. He was trying to verify his record readings at La Parida. The El Pao test confirmed his mathematics. Then the readings and calculations were sent to Dr. Louis Slichter (Sumner Slichter is his brother), Director of Geophysics at the University of California, for verification. His notation came back that everything was correct but perhaps someone had gotten the decimal points in the wrong place. But the decimal points definitely were in the right place and La Parida began to look more and more a champion.

By early 1948 La Parida had its name changed to Cerro Bolivar, and drillings had proven some 165 million tons of reserves. Seven diamond and two churn drills were operating, and 47 holes had been drilled to an aggregate depth of 11,436 ft. Moisture content of ore samples was 10 pct, indicating an average natural iron content of 56.7 pct. Most of the holes bottomed in quartzite, the average ore depth being 148 ft. The area covered by exploration was 18,000 ft long with an average width of 1200 ft. An exploratory tunnel had been driven, and the specific gravity of the ore was 12 cu ft per ton. A railroad survey was started, and river studies were planned to investigate dredging possibilities. Other ore showings in the Cerro Bolivar (La Parida) area—Arimagua, Rondon and Altamira—were given a quick look and estimates made that they also were large deposits, having perhaps an average dry analysis of 63.28 Fe, 0.113 P, 2.29 Si, 0.10 Mn, 1.53 alumina, 0.135 Ti, and 0.005 S; loss on ignition 5.31.

During the 1947-48 period also, interest had grown in a deposit at La Grulla, which was back over in the usual iron area east of the Caroni, in Federal Reserve territory, about 30 miles northeast of Bethlehem's El Pao mine, 30 miles south of the Orinoco River and 90 miles due east of Ciudad Bolivar. The ore seemed to be in relatively small bodies, but high grade. A concession was petitioned for, as the ore was hard and crystalline, of about 65 pct Fe with low-phos, and would make excellent lump ore. The individual ore bodies could, perhaps, be mined as a district, and reserves of 10 million tons were indicated. By late 1947 a magnetometer report over one deposit was on the order of the spectacular responses on La Parida hill. The high-grade ore showings on this deposit were very impressive, particularly when their extension in depth was indicated by the magnetometer interpretations. Several other deposits appeared connected as the result of geophysical work and there was some similarity between the magnetic responses and those at El Pao. By June 1947, five concessions

Sailing craft on the Orinoco near Palua. Venezuela hired ex-Maj. Gen. Tyler of U. S. Army Eng. to survey river with object of getting 35-ft. draft ocean ore carriers up the Orinoco as far as Palua.

Cerro Bolivar, the greatest and richest mountain of iron ore in the world, was discovered in April 1947. The explorer's camp shown here is perched on lumps of iron ore of almost maximum theoretical purity.

The "Spindle" anchored in the Orinoco at the headquarters town of Ciudad Bolivar. A barge-load of cargo has just been brought from the Piacoa camp.

Cerro Bolivar's climate is hospitable, the sunsets are stupendous, and the surrounding savanna terrain pleasing to the eye. This view north from Cerro Bolivar shows a road cut to bring in Longyear diamond drills.

were obtained in the La Grulla area. However, by late 1948 La Grulla began to be somewhat disappointing. In some places drillings showed that the hard, crystalline ore in outcrops did not extend more than a few feet below the surface. Further, some of the subsurface connections between surface outcrops did not appear to exist. However, drillings were continued.

In early 1949 over ½ billion tons were proved at Cerro Bolivar by drilling. La Grulla continued disappointing despite extensive drilling, with only 40 million tons proved up.

As of the present time, Mack C. Lake is in full charge of the Venezuelan operation for the Corporation, and has been made the first president of the newly-formed subsidiary, Orinoco Mining Co. R. J. Wysor, former president of Republic, has for some time been associated with Mr. Lake on the job as a consultant. Mr. Lake has set up offices at 25 Broadway and is energetically building up a staff. It is the intention to bring at least 10 million tons out annually by late 1953.

In the Cerro Bolivar area south of the Orinoco and west of the Caroni, U. S. Steel has 18 denouncements of 100 yr duration with probable reserves of the highest grades of iron ore greatly in excess of 1 billion tons. Some idea of the size of the Cerro Bolivar deposit can be obtained by comparing its linear dimensions with the world-famous Hull-Rust-Mahoning open pit at Hibbing, Minn. (see drawing) now scraped almost down to Taconite bottom. Cerro Bolivar alone contains more tonnage of iron than Hull-Rust ever produced or ever can produce.

There are other probable deposits in the Cerro Bolivar area but the Venezuelan government has closed the territory against further denouncements and only concessions of 40 yr may now be obtained.

U. S. Steel also has six concessions (La Grulla, etc.) in the usual area east of the Caroni and south of the Orinoco, which contains a wide variety of ores and respectable tonnages, although in no way comparable to Cerro Bolivar. These all are on the Federal Reserve and operation for a term of only 40 yr is possible.

The Corporation must soon make a decision on how it plans to bring its large tonnages at Cerro Bolivar out of the country. A railroad survey has been made straight across country from Cerro Bolivar to a tidewater terminal on the north coast near Barcelona, a distance of 274 miles (see map). This will require a bridge across the Orinoco at or near Ciudad Bolivar, about 4.5 miles in length including approaches. Pay load per train would be 10,000 tons and the annual tonnage 15 million tons. The total cost will be on the order of $113 million. At the Barcelona port there is excellent deep water so that the biggest ocean ore carriers can be brought in for loading.

Another alternative would be a railroad for 91 miles from Cerro Bolivar to a river terminal at the confluence of the Orinoco and Caroni Rivers. Then the Orinoco must be dredged to permit bringing up the ocean ore carriers to load at this terminal. The 91 miles of railroad would cost $51 million. The river survey contemplates a channel of 34 ft from the confluence of the Orinoco and Caroni, down the Orinoco and Macareo Rivers to a point

60 miles from tidewater, and from there a gradual deepening of the channel to 38 ft at sea level. To do this requires dredging 45,254,000 cu yd of bottom, at an estimated cost of $18 million. Annual maintenance would be $1.1 million. A combination 34-38 ft. channel would open up the river for even the enormous ocean carriers planned by the Corporation. The problem of dredging and maintaining a river of this size is a tremendous undertaking for a private company. In the United States the government would do the job although it has not yet been determined to what extent Venezuela will participate.

The distances to U. S. consuming centers by the various routes are as follows: Cerro Bolivar to Orinoco River port = 91 miles; Orinoco River port to tidewater = 170 miles; mouth of Macareo to Baltimore = 2280 miles; Baltimore to Pittsburgh (via B. & O. or Western Maryland) = 334 miles; mouth of Macareo to Mobile = 2460 miles; Mobile to Birmingham = 276 miles (rail) and 280 miles (water); Cerro Bolivar to Barcelona = 274 miles; Barcelona to Baltimore = 2190 miles; Barcelona to Mobile = 2190 miles.

A price comparison might now be in order, between U. S. Steel's Venezuelan ore, Labrador ore (assuming the St. Lawrence Seaway), Superior Taconite nodules and current non-bessemer ore shipments from the Mesabi. The following table has been constructed to this end, based on a variety of informed estimates. In the table L.E. = Lake Erie price (1949) based on $7.20 per ton for Mesabi non-bessemer ore.

								1949 Basis	
	Dry Iron	Nat. Iron	Phos.	Si_2O_2	Mn	Nat. Mn	Moist	L.E. Price	Per Unit Fe
Labrador Ores ..	59.3	53.4	0.071	6.76	1.51	1.36	10.0	$7.72	14.5¢
Venezuelan Ores.	63.8	59.0	0.113	2.29	0.10	—	8.75	8.54	14.5¢
L. S. Taconite Nodules	65.4	65.0	0.025	6.75	0.10	—	0.5	9.40	14.5¢
Superior Natural Ores (1947).	58.2	51.1	0.064	7.98	0.70	—	12.23	7.14	14.0¢

It will be noted that even at Lake Erie ports, Venezuelan ore shows an equivalent unit cost with Superior Taconite nodules and Labrador ore. Natural ore from Superior, however, shows up best of all, as might be expected. However, natural ore is a disappearing metallic which has no bearing on the economic relationship between future iron supplies.

Also to be considered is that the cost relationships in the table are modified by the iron units per ton of material passing through the blast furnace. Furnace operators have of recent years recognized that distinct savings in hot-metal costs are secured by using sized ores with larger proportions of coarser material in the form of concentrates or agglomerates. The higher grade ores produce less slag and thus require less coke and limestone. The coarser structure burdens usually reduce more rapidly, thus permitting larger metal production from the furnace; flue dust, with its content of fine ore, is also reduced. It is thus apparent that ores should be evaluated as to their effect on the cost of producing hot metal.

With this in mind, and thus allowing proper credit for structural and chemical characteristics, Venezuelan ores and Superior Taconite nodules improve slightly in cost factors.

From a hot-metal cost angle, and taking a rather brave shot into the 1960's, the Superior

Boccardo Hill, above Bethlehem's El Pao concession. Standing on lumps of 68 pct Fe (L to R) Jesus Lareto, Venezuelan Engineer; M. D. Meixel, El Pao foreman; Wm. Rankin, head of Bethlehem's Venezuelan operations; Mack C. Lake, the first president of U. S. Steel's Orinoco Mining Co.; and Mr. Boeckman, Esthonian engineer.

These men recorded the world's highest magnetometer readings. (L to R) Jose Ramon Zurita, Venezuelan trained to operate Tiberg magnetometer; Wm. Moyka; Sam Sargis, a magnetometer expert borrowed from Geneva Steel Co.; and Fred Wright.

Diamond drillers at fabulous Cerro Bolivar occupied these two tents surrounded by great chunks of iron ore.

Axel Auguston ran the field force of diamond drillers for E. J. Longyear Co. His wife Betty (right) cooked for the crew. Here she is with her helper before the dining hall at the Piocoa concession.

nodules should show up well at all the Lake consuming centers and down into the Youngstown area. Venezuelan ores will dominate the Gulf area, the East Coast, Pittsburgh and possibly work up into Youngstown. Labrador ores, via the St. Lawrence Seaway, will show up well in the Lake Erie consuming centers. Of course, the actual proportions of each metallic used at any producing point will be dictated in part by the type of material being developed by the particular companies.

By looking ahead to the late 1960's, it is possible to estimate that the steel companies other than U. S. Steel and Bethlehem most likely will be drawing ore from Labrador at the rate of 10 million tons annually, will have at least 17 million annual tons of Taconite-concentrate capacity in operation, be pulling a ½ million tons from Liberia, and be buying some 4 million tons of Venezuelan ore. Dependence on Lake Superior natural ores will have practically disappeared. The capital investment to finance such a program will exceed $600 million, exclusive of the lake vessels required for Labrador ore.

The U. S. Steel Corp.'s ore balance sheet in the late 1960's will probably show at least 15 million annual tons coming from Venezuela and 10 million annual tons from beneficiated Lake Superior Taconites. U. S. Steel definitely plans to pull 500,000 tons of Taconite nodules out by 1952. This will cost that Corporation in capital commitments a sum just about as much as for the other steel companies, $600 million plus.

The American steel companies, therefore, must get their hands on over $1.2 billion dollars to sink into new capital investment in the period up to 1965, and this staggering sum is in addition to the normal rebuilding of steel-producing facilities. It's going to take a lot of doing —some good profits and very friendly bankers! If the ore program is followed through, however, the drain on Superior natural ores will be lifted in time to retain possibly 500 million tons as a strategic stockpile. If there ever must be another war, just possibly it, too, will be fought in great part off that old standby—the Superior natural ores.

The next few years will see a dynamic stir in the steel industry having no equal for decades —certainly not since flat-rolled capacity fell before the onslaught of the continuous mill. But that was an intercompany competitive struggle, whereas today's ore program is a matter of company and country survival.

Certainly the U. S. Steel Corp. offices are already surging with plans to keep its construction program in balance. The Venezuelan ore fields will be mechanized as much as human ingenuity can devise, the railroad must be built or the Orinoco dredged to get the ore out, and a $50 million boat program must get under way to pick up the ore when it arrives at tidewater. It is planned to use giant ocean carriers, 45,000-ton jobs rivaling the battleship Missouri in size, to shave down transportation costs. (As comparison, one of the largest of Lake carriers, the Irving Olds, barely squeezes over the 26 ft sill at the Soo locks with 18,000 tons.) The first 2 million annual tons of Venezuelan ore should certainly go to Mobile and up the Warrior River to T. C. & I., where blast furnaces are now literally agasping for rich ore. Production may rise there as much as 15 pct with the same furnaces. Over ½ million tons of hard ore and many millions of tons of blast-furnace grade will start coming into Baltimore (where already bulk unloading facilities are available) for rail shipment into Pittsburgh via the B. & O. and W. Md. And, most interesting, 1.5 million tons would go into the U. S. Steel Corp.'s Eastern steel mill, when built. For while the Corporation is designing all the equipment for Venezuela, it is also probably designing a large size steel mill for its site near Trenton, N. J. The total cost of such a manufacturing plant would run into some hundreds of millions of dollars.

It isn't necessary to be privy to U. S. Steel Corp. secrets to make some estimate as regards the Trenton plant. Press comment has been busily building the plant 10 yr from now, but obviously the desired planning would be to target its completion to coincide with adequate ore from Venezuela, and that certainly is before 1955.

The finished steel market in the Trenton-Philadelphia-Washington area is a lush one, some 7 million tons yearly, which holds up well even in hard times. It's a diverse market—over 1½ million tons of sheet and tin plate; 700,000 tons of wire; and 600,000 tons of plate. Bethlehem takes care of 40 pct of the market and most of the remainder comes in from U. S. Steel and other steel companies in Pittsburgh, Youngstown and Chicago. The Corporation can no longer afford not to replace wearing-out Pittsburgh tonnage with lower cost tonnage in the east to service this market.

A Trenton plant of 2 million annual ingot tons would require three blast furnaces, and, say, two openhearth shops of eight 150-ton furnaces each. The plant would run on strictly an ore burden with mill-return scrap. No bought scrap would be used. There would be at least one hot and one cold continuous mill, a wire mill, probably even pipe and bar mills. The whole thing would certainly be a long, level, straight-line affair, from blast furnaces to shipping dock, quite possibly the most beautifully designed integrated mill in the world. When both the Venezuelan ore and the finished steel market in that area are viewed together, it would be expected that by mid-1960 another "South Chicago" will have sprung up on the flat farmland beyond Trenton, N. J. The Delaware River already has a 41-ft channel up to Philadelphia, and the Army plans to scratch out a deep channel on up to Trenton, thereby permitting ocean ore carriers to dock alongside the steel mill (see map). Or, on occasion the ore can be barged from the unloading docks at Baltimore, through the C.&D. canal (dotted line, map), and thence to the steel mill.

For 60 yr the steel industry has viewed its ore problem with all the bland indifference of the heavily endowed. But with the endowment about gone, it is apparent from all the foregoing that the challenge has unleashed a great dynamic drive from Hibbing to Venezuela, from Labrador to Liberia, from Trenton, to Pittsburgh to Birmingham. It all involves enough imagination, ingenuity and romance to quicken the most sluggish pulse. Steel men are long going to remember the 1950's.

Where Can Coal Go from Here?

AN analysis of the bituminous coal situation by an authority who traces the production, mining, safety, markets and labor trends in comparison with other fuels.

by Howard N. Eavenson

BEFORE 1918 the production of coal experienced a steady growth, with slight recessions in ouput in only twelve years (1837, 1841, 1852, 1855, 1861, 1874, 1876, 1889, 1894, 1905, 1909, 1914), and in each of these cases the loss was recovered in the following year and the increase continued. In each decade the output was doubled. Prices, while not high, allowed some profit. In 1918 during World War I, an output of 580 million tons was reached and with the exception of 1923 this total was not again reached until 1942, in World War II. Losses of output in this period were much greater than ever before; in 1932 the total was 270 million tons less than in 1918 or 1923, and the loss after 1923 was not recovered until 1942. The greatest output was 631 million tons in 1947.

The per capita production, which is practically equal to the consumption, reached its maximum of 5.6 tons in 1918, higher than it has ever been since, even in World War II. In 1930 it was 2.5 tons and now it is about 4.0 tons per year.

During recent years, taking the period 1935-39 as a base, coal's output increased 48.5% in 1948, crude petroleum 72%, fuel oil 97%, and natural gas doubled.

While many of the modern mines are giants they are only a small part of the total number working. In 1918 the total number of mines reached a peak of 11,038, in 1932 only 5427 of these or 49% were working, but the average production had risen to 57,100 tons. In 1947 the next maximum of 8700 mines was reached, and today there are less than 8000 mines.

In 1918 the average mine production was 52,490 tons, in 1947 it was 72,500 tons. In 1918, the 821 mines producing over 200,000 tons annually mined 48.5% of the total, while in 1947, the 801 mines of this class mined 66.1% of the output.

In 1919 there were 6636 producers operating

8282 mines, and the number of both items increased greatly until 1923. Following 1923 the number of producers decreased until 1934 when it was less than 4900. With increasing outputs and World War II, both number of mines and producers grew, and in 1947 there were 8014 producers of which 270 companies mined over one million tons each or 57% of the output. In 1909 forty-three producers mined over one million tons each or 18.9% of the total output. It is evident that while the number of operators and the output fluctuates widely the number of large units and their percentages of the total has been steadily increasing.

The productive capacity of the mines in 1919 was 621,000,000 tons in 260 working days; this increased somewhat until 1923 and then decreased until the outbreak of World War II; now it is about 702,000,000 tons on the same operating time, but this can be increased by operating six days per week at an increased cost. Much of this growth is due to the large numbers of small truck mines and strip mines which do not prepare their coal; many of these have already ceased operating, and more will do so under competitive conditions.

Mr. Eavenson is a consulting coal mining engineer, having his own firm, Eavenson and Auchmuty, in Pittsburgh. He is a Past President of AIME.

Machinery—Most people know that the coal mines for many years have been adapting machine methods to the various mining operations as fast as they have been developed until now most mines, where the seam conditions allow, are cutting, shooting, and loading the coal by machines which greatly reduce the manual labor required. The ultimate development of machine mining at present is reached in the large strip mines where the overburden, sometimes as high as 100 feet, is removed by immense power shovels, some with shovels taking 50 cu yd at a time. The coal is loaded into trucks handling from 15 to 30 tons, taken to the tipple and prepared in the same way as the coal from the underground mines. In 1919 the output of strip mines was 5,635,000 tons and came from 13 states. In 1947 the strip mine output was 139,400,000 tons produced in 24 states and was 22.1% of the entire production of bituminous coal. Pennsylvania was by far the largest producer of this type of coal, with West Virginia, Illinois and Indiana following in that order.

Due largely to the increased use of machinery the output per man-day has steadily grown since 1919. In that year 4.64 tons were produced per man-day worked underground in the deep mines while in 1947 the corresponding figure, but for a shorter day, was 5.49 tons. The strip mines, however, in 1947 averaged 15.93 tons per man-day, and many did better than that, making the total average of all kinds of mining 6.42 tons.

Another great change around the mines in recent years, and proceeding rapidly now, is the use of mechanical cleaning and preparation. In 1919 hand picking the impurities from the coal in the larger sizes was practiced, with the smaller ones generally going into the car without further preparation. In modern plants no hand picking is done. The coal over ½ inch in size is all mechanically cleaned and some of it heat dried, and the quality of each size is as nearly uniform as it can be made from a natural product at a reasonable cost. Much of the smaller sizes is also either dry or wet cleaned. The use of loading machines has been one of the causes of this, and the market requirements furnished the rest. Such plants are expensive, ranging from a half million dollars for a small one to as high as 10 to 15 million dollars for some of the large ones being built in this region. The necessity for such plants is one of the main reasons for the rapid and large increase in the number of producers having large outputs.

Wages—Much has been said recently about miners and another increase in wages. Data about actual earnings in the past are scarce, but in the report of the Coal Commission the earnings of all classes of employees in 1921 are given, from which the average for the year was computed as $729.38; the running time was only 149 days. At that time the hourly earnings for inside laborers—the lowest rate—was 64.2¢ per hour in Pennsylvania, the lowest in the northern fields and 62.4¢ in West Virginia. The earnings for outside laborers was roughly 10% less than this figure. In 1924 when the Jacksonville agreement was signed the average earning in Pennsylvania was 74.4¢ and in West Virginia 54.3¢ per hour. The rates and earnings both decreased until, in 1933, the inside laborer in Pennsylvania earned 38.8¢ and in West Virginia 32.9¢ per hour, with less working time. This was the nadir of earnings in modern times, and there are no data comparable to those above for 1921.

Under the present scale for the year from July 1, 1948 to June 30, 1949 the average earning per hour was $1.938, the average weekly one was $67.88, varying between $75.97 in July 1948 to $56.03 the following June, the decrease being due to poor business and the three-week shutdown in April and June. The average earnings for the year were $3228.82. These figures cover 67,000 men in the southern mines—about 60% of the total membership production. No other data are available for this period but normally these are roughly equal to those of the entire country. None of the "fringe" payments, such as for vacations, social security, unemployment, etc., are included, and there were no such payments in 1921. The sum of these payments amounts to about $1.75 per day or about 25¢ per hour.

So far as we know no other competitive and basic industry pays such high weekly earnings as these. In this industry 60% or more of the cost of the product is due to labor.

Much has been said in the past about company towns. In 1945 two thirds of the miners lived in their own homes or in houses rented from others, and one third in houses owned by coal companies. In the north the proportion of owners of homes

is higher than this; most of the mines in the south are working under leases which do not permit the sale of houses owned by the lessees. Few coal companies now operate any facilities not needed directly for their business, and a few have even sold their stores. Miners now have living conditions fully as good, if not better, than the average of the communities in which they reside.

Good roads and automobiles, as well as policy, have caused the company stores to be fully competitive with their neighbors, and during the many recent strikes the families of the miners have largely lived on the credit advanced by these stores.

At the time of the Coal Commission the denial of civil liberties to the miners was the subject of much study; if there is any such denial now in the coal towns it is certainly not the miners who suffer it.

In the press and the communiques issued by the head of the UMWA the impression is always given, even where the direct statement is not made, that the union, assisted by the U. S. Bureau of Mines, was the only party about the mines who had any interest in the safety of those working in and around them, and that they received little or no help from those owning or managing the properties. This is decidedly not the case, as many mines had active safety programs in use before the bureau did more than report accidents, and it has only been within a comparatively few years that the union took any official part in this work. Any extractive industry has more hazards than a manufacturing plant and because of mining conditions it is impossible to give as close supervision as can be done in a shop or building work. There always have been accidents in mining work, and there always will be some, even if and after human nature has changed greatly. The coal mining industry has no reason to be ashamed of its record in this respect.

Records of all accidents occurring in coal mines are reported promptly to both state and national government and are later published. These past records have been recomputed by the U. S. Bureau of Mines to the present basis of portal-to-portal time, and are comparable with the published data before that time. Since 1930, and using that year as 100, the fatal accidents have steadily declined to 42% or from 3.46 to 1.46 per million tons in 1948; in the same period the nonfatals have declined to 46% or from 152.3 to 70.2 accidents per million tons in 1948 including all injuries of every kind where the employee was absent the day after the accident occurred—a very rigid standard. On the basis of man-hours of exposure the fatal accidents declined from 100 in 1930 to 64 in 1945, and the nonfatals from 100 to 72.7; no data are yet available after 1945.

Daniel Harrington was in the health and safety division of the Bureau of Mines—part of the time its head—thirty years before he recently retired. In a recent radio broadcast he said "that 75% to 80% of all mine accidents are due to human failure, like those in the home or on the highway. The biggest hazard in coal mining was due to the miners' carelessness—his frequent failure to protect himself." Harrington is one of the best known safety men in this country, who knows what he says and is not afraid to say what he thinks.

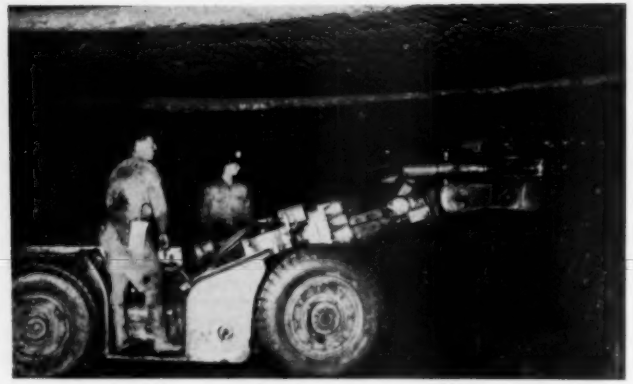
The main causes of all underground accidents are from falls of roof and on haulages, where only continued personal care and interest can reduce them, and few operators have noticed any unusual improvement since the UMWA became officially interested.

Markets—Markets have always been one of the coal industry's chief troubles, and also the cause of most of the labor troubles which have afflicted it, and these have changed materially since 1920. Then the railroads were the largest consumers, using 26.6% of the total. In 1948 they used 18.7% of an output slightly over 22 million tons larger; the use of power utilities has grown from 6.0% to 18.0% now; coke ovens and the gas industry changed from 15.0 to 20.4%; the amount taken by retail dealers fell to 16.9% in 1948, no comparable figure being available before 1934 when it was nearly the same tonnage, but 25.0% of the total used. This includes some coal, although not much, used by small industries, and is affected greatly by the weather. In 1944 the total taken by retailers was 124.9 million tons or 21.0% of the largest total production we have ever had, plus imports less exports. This market decreased to 99.1 million tons in 1947. The prices obtained for the coal sold to retailers are the largest source of profit the industry has.

In 1935 the U. S. Bureau of Mines began to publish data about the amounts of each variety of fuel used for domestic and commercial purposes. At that time the amount of heat in Btu furnished by bituminous coal or its derivatives was 54.9%, fuel oil and petroleum coke was 12.7%, and natural and bottled gas 9.3% of the total. In

Wide World Photo

Not everyone agreed with John L. Lewis—including non-union coal miners at Clearfield, Pa., who were guarded by a man with a deer rifle during last September's strike.



Mechanization, exemplified by this universal self-propelled cutter, was hastened by increasing labor costs. The continuous miner, inspired by coal cutters, will increase the tons per man-shift.

1944, the year of the largest use of coal, its share was 52.5%, fuel oil 17.1%, and gas 12.5%; in 1947, the year of the largest production of coal, its share had fallen to 42.0%, fuel oil increased to 25.7%, and natural gas to 17.9% of the total. This relative increase in the use of the liquid fuels for these purposes increased steadily during 1948 and in this year. In all of the new housing projects in the country about 73% of the units will be heated by fuels other than bituminous coal. It is very difficult now to find a manufacturer of heating appliances who will show any interest in a new coal heating or cooking unit, no matter how much better it is than the types in use.

If this trend continues, and it has been accentuated by the numerous stoppages of coal output, it will make a decided change in market trends. The southern mines have been furnishing the larger proportion of steam coal, and when the income from other sizes must be increased to compensate for the reduction in domestic sizes it is questionable if they can ship steam coal into the markets it now has, regardless of quality.

Constant study is being made for new outlets for coal. Bituminous Coal Research, Inc., is designing better heating and cooking units and anything else that will improve the value of coal to consumers under a fund raised by voluntary subscriptions, amounting to about a half million dollars a year, totally inadequate for such a basic industry but still a good start. Its locomotive development committee, sponsored both by coal producers and railroads, has designed a coal burning gas turbine locomotive, and two of these will begin experimental runs during 1950. Coal heating service is another way the industry is trying to promote the better use of coal in house heating. The use of coal in manufacturing chemicals is steadily increasing. The gas turbine may replace the Diesel engine in the future. The largest increase in coal consumption in prospect is its use in making oil; when this is done, if and when the supplies of petroleum and natural gas begin to show signs of material depletion, an increase in coal production will be necessary.

Another factor, of much greater importance to coal than to the liquid fuels, is the constantly increasing freight rates on the railroads. In 1928 the average freight rate per ton of bituminous coal was $2.27, in 1948 it was $2.74 per ton. In 1942, for the first time since World War I, the average mine realization of $2.36 per ton passed the average freight rate of $2.31. The liquid fuels move by pipe lines, and the proportion of labor to total cost is much less than for railroads, as is also the case in producing liquid fuels.

For some time the news reports have been full of statements about the immense profits in the coal industry and its ability to absorb unnamed increases without any price increase. While some mines have made earnings, and some losses, in each year since 1928, the whole industry has shown a loss each year from before 1928 to 1939, since which time it has shown a profit after taxes each year to and including 1946, when the amount on the reported capital invested was 4.55%. In the same year total manufacturing earned at the rate of 8.74% on invested capital, and in every year since 1925 its earnings after taxes have been higher than those of the coal industry.

The worst trouble in the industry has been its labor relations, due to its many units, local mining conditions, overdevelopment and other causes. Before 1927 most of the mines were nonunion, even in the northern fields, but this was all changed with the advent of the New Deal, the Wagner and Wage and Hour Acts, which unionized all but a small part of the mines, about as they are now. With the ability of one leader to control nearly the whole of a basic industry, the economic power of the union has grown tremendously as has been demonstrated frequently during the past few years. This year we have had a two-week "memorial" period to protest against the appointment of a director of the Bureau of Mines, a one-week period in June to reduce the stock piles, the three-day-a-week period from July to September to equalize the work, a no-day-a-week period only recently ended. We have seen collective bargaining turn into collective bludgeoning and finally a seven months' negotiation during which not even the wants of the union

have been disclosed excepting that they will only amount to 30 or 35 cents per ton, which on the usual proportion of stated to actual figures probably means 50 to 60 cents per ton.

In 1943 Secretary Ickes forced the industry to accept a national wage agreement, much against its wish. It was signed in November 1943. A renewal of this was made in 1945. In 1946 the operators rejected the demand of the union for a welfare and pension fund, but Secretary Krug made such an agreement while the mines were under his control, the pension fund being 5 cents per ton; in the agreement made in 1947 this amount was increased to 10 cents per ton, and July 1, 1948 saw this amount made 20 cents per ton in an agreement extending to July 1, 1949 under which the mines are now working. The Federal Mine Safety Code was included in one of the government's agreements. In the 1947 contract was first inserted the clause that the men would work "during such time as such persons were able and willing to work." It has been stated by one of the operators' negotiators that this clause was only discussed by the Sub-Scale Committee, and that its purpose "was to prevent or avoid suits for damages against the general treasury of the UMWA resulting from local strikes occasioned by irresponsible local unions or irresponsible men or *agents provocateurs*." As interpreted now it will destroy all prerogatives of management. He further said, "The operators have had only two real considerations in their collective bargaining agreements heretofore: (1) the mutual obligations of operators and miners to live up to and fairly administer the contract to maintain production, settle grievances, and stop strikes during the period of the Agreement; and (2) the right to manage the mines as clearly set forth in the Agreement. These two provisions were all-important to management. These are now nearly eliminated."

One other point of difference now is the management of the pension and welfare fund. The agreement was that the miners would elect one trustee, who is John L. Lewis; the operators one, who was Ezra Van Horn, now being succeeded by ex-judge Charles I. Dawson. These two were to select a so-called neutral trustee who is now Senator Styles Bridges, now in process of resigning. His neutrality may be judged by the fact that until the fund was nearly exhausted all his votes coincided with those of Lewis, excepting the last one, suspending the operation of the fund. The operators want a definite system of disbursing this fund with its operations, those receiving payments and how much, etc. known to all. Now, while the operators furnish *all* the money none of it is paid to anyone in the supervisory, auditing, or engineering force, but only to a member of the UMWA who has worked in some mines twenty years and is sixty-two years old. The whole system is extremely loose and unfair.

The following opinions, or thoughts, on possible remedies for some of the present troubles and for those which will arise as soon as real competitive conditions exist again, which will likely be sooner than many think, are my own, and not necessarily those held by most of the managers in the industry.

For the short period I would suggest: (1) That the antitrust laws be extended to apply to labor as well as to capital with the same penalties applying to all. This would automatically eliminate national agreements and prevent any labor leader controlling absolutely an industry whose stoppage may endanger the nation. (2) That all pension funds in industry be administered by totally disinterested trustees, acting under laws similar to those governing insurance companies. (3) That such pension funds begin now, but allowing such retroactive arrangements as may be agreed upon by the parties interested. (4) Stop the collection of all "check-offs" for union dues or funds. These changes would considerably brighten the possibility of better labor relations.

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Modernity in coal mines is typified by this continuous cycle rotary dump at the Old Ben Coal Co's. No. 9 mine in West Frankfort, Ill. It automatically uncouples cars, tips them upside down to empty them onto the shaft cage for the trip to the surface. Steel beams, reinforced concrete ceiling, and sturdy block walls are other features.

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Another matter which handicaps coal in competition with the liquid fuels is the tremendous differences in the way the two classes of fuel are treated by the Income Tax Bureau. All are extractive industries and should be treated the same way as far as depletion charges are concerned. Because of large increases in the costs of new plants and equipment, coal, as well as all enterprises affected the same way, should be allowed to charge accelerated depreciation and also —a charge for obsolescence due to changes in methods of mining and depreciation.

It was suggested a short time ago, I think inadvertently, that one possibility for improving the economic situation for the coal industry was to move portions of it. As we must mine the coal where it is, with all its varying conditions and qualities, which fit some seams for different uses than others, and as 80% or more of the total is mined and used east of the Mississippi River, the regulation of equal outputs or equal running times is likely to be a task transcending the powers of our most cartooned labor leader, to say nothing of anyone else. Free competition can only result in reestablishing the conditions existing in the early thirties, less the restraining influence of the union contract.

One possible solution would be to extend the proposed Brannan law to coal; if the operators could mine all the coal they wanted to, sell it for any price they could and be paid by Uncle Sam enough to insure an equal income for the proposed period, everyone would be happy, high wages could be paid, and all could have pensions —for a while.

To meet the competitive conditions certain to come soon, it is hard to propose any solution acceptable to a large share of the industry. It will be necessary to impose minimum prices, and if the industry could legally be organized into districts, including all producers in each one, and be administered by the operators, but subject to control on policy matters by a federal board, it would approach a satisfactory solution. Such a scheme would eliminate the delays occurring under the Guffey Act, and while it would control output to some extent, some such control is a necessary remedy.

From a national standpoint there should be a fuel policy, which could allot the use of each of our fuels to that best suited for it. Even the oil men deprecate the use of oil under manufacturing plant boilers, and natural gas should be reserved for cooking, house heating, and other uses to which it is best adapted. Such a policy was used during the war, and if it can be done legally it would greatly influence the conservation of our fuel supplies.

Another and still longer look from a national view, is the need of a policy to reduce the amounts of our fuels left in the ground in the extracting process. Speaking only for coal, for economic reasons almost entirely, we have not recovered over 50 to 70% of the coal in the seams mined, and this proportion is tending to increase rather than to decrease. We should formulate a scheme where those operators increasing their percentages of recovery would be subsidized by the government. It would not cost as much as the amount paid to the potato growers and others who raise a crop each year while the one the coal operators mine occurs only once.

Books for Engineers

Guide to the Coalfields, 1949. Edited by R. H. Walkerdine and C. T. Jones. Colliery Guardian Co., Ltd., 30 and 31 Furnival Street, Holborn, London, E.C.4, England. 436 pp., maps. 16s.9d. The major portion of this volume consists of condensed descriptions of deep coal mines in Great Britain. The type of coal, number of men employed, power supply used, and the name of the manager are given. Maps show the location of the pits. Related technical societies and organizations connected with the coal industry are dealt with briefly.

Geology: Principles and Processes. By W. H. Emmons, G. A. Thiel, C. R. Stauffer and I. S. Allison. 3 ed. McGraw-Hill Book Co., New York, Toronto, London, 1949. 502 pp., illus., diagrs., charts, maps, tables. $4.50. In simple but technical style, this book presents the fundamental concepts of physical geology and gives a scientific view of the processes that operate on and in the earth. Special features of this third edition include new emphasis on the interpretation of landscape and geologic structures as seen from the air; less emphasis on metamorphism and vulcanism; increased attention to the topography of the continental shelves; an outline of the nature of matter and a new discussion of weather maps.

Geology for Engineers. By J. M. Trefethen. D. Van Nostrand Company, Toronto, New York, London, 1949. 620 pp., illus., diagrs., charts, tables, maps, 9¼ x 6 in., cloth, $5.75. This book presents certain fundamentals of geology which will provide engineers with the basic knowledge they need for their work and to enable them to use the geological investigations of others. Courses in physics, mechanics and surveying are required prerequisites. Geological structures, subsurface water, earth movements, and shorelines are among the important topics considered.

Uses for Radioactivity. An extensive bibliography on the industrial uses of radioactive minerals has been prepared by Arthur D. Little, Inc., and is available gratis from the Company, at Cambridge, 42, Mass. In view of the firm's belief that greater utilization of radioactive tracing techniques is hindered by a lack of knowledge of the subject, the bibliography should prove useful.

Books may be purchased through I. P. Klein, Book Department, AIME. Whenever possible 10% discount is given. Order government publications direct from bureau concerned.

Differentiating Characteristics of

The Engineering Curriculum

An evaluation of the function of the engineer, so that means whereby education can best serve his needs can be adopted.

by S. C. Hollister

THERE has been a steady increase in specialized branches of engineering during recent years. Some of these are simply specializations in the function of engineering, such as welding engineering, tool and die engineering, petroleum engineering, and refining engineering. Others are associated more closely with the basic sciences, as for example, geological engineering or engineering physics. Still others tend toward the border line of business, such as administrative engineering or general engineering.

Many engineering schools have catered to the popular demand for specialized training in a wide variety of engineering specialties. As a result there is now some confusion as to what really characterizes an engineer, or indeed a curriculum suitable for training for any of the branches of engineering. It appears timely, therefore, to examine this situation and to prepare, if possible, a compact statement of what characterizes *engineering* training. In fact, if accreditation is to be successful, curricula so accredited must contain such characteristics as may readily identify them as *engineering* curricula.

Engineering education is surrounded by other educational operations. It will be helpful to explore the boundaries between engineering and those contiguous fields in order that a clearer understanding of engineering education may be had, and that we may attempt a direct definition of the characteristics of an engineer.

It is generally accepted that engineering training is at the college level. Thus one segment of its boundaries would be represented by the trade school, the technical institute, and the general secondary school. Within the collegiate area it is bounded by programs in the liberal arts, in science, in business, architecture, agriculture, etc. Boundaries between college and secondary schools or trade schools are well defined. Similarly the technical institute operating at post-high school level is readily distinguished from engineering training at college level. Confusion arises in varying degrees between engineering training and other college work, particularly in curricula which are divided between engineering and other college subjects. I will not undertake to evaluate curricula which are in part engineering and which are heavily charged with work in business, liberal arts, or some other field. Such courses are highly desirable and more of them should be given. The sole problem considered here is whether such curricula should be considered as *engineering* curricula leading to an *engineering* degree. This calls for a specification of differentiating characteristics contained in an engineering curriculum.

The author is Dean of the College of Engineering at Cornell University, Ithaca, N. Y. This paper was presented at the annual meeting of ECPD, Chicago, October 28, 1949, and adopted by the Committee on Engineering Schools.

A comparative study of curricula in science programs with those in engineering reveals a significant difference between them. Engineering programs characteristically include mechanics, mechanics of materials, and courses in the properties of materials. It is also characteristic that in varying degree there will be courses relating to structures, apparatus, or machines, and the principles upon which they are designed, constructed, and operated. Such courses are not found in the science curricula. For example, a curriculum in mining engineering will include mathematics, physics, chemistry, and geology as will also a curriculum in geology. But the mining engineering curriculum also contains courses in mechanics, hydraulics, hydraulic machinery, heat

engines, and electrical equipment. The inference is that the mining engineer is going to design, construct, and operate and that the geologist is interested in the formation and nature of the earth's crust but that he is not intending to design, construct, or operate works either above or below ground.

When curricula in engineering and in architecture are compared, courses in mechanics and materials and sometimes in structural design are found in the architectural program. The principal characteristic of the architectural curriculum, however, is training to achieve artistic excellence in the proportions of parts and in the decorative treatment of structures. In practice the architect emphasizes function and beauty, whereas the structural engineer emphasizes function and safety. The two fields obviously lie very close together and it is conceivable and, in fact, is occasionally true that the architect may act as a structural engineer, and vice versa.

A comparison of curricula in electrical engineering and in physics reveals the presence in the engineering curriculum of mechanics and materials of engineering. The physicist ascertains the behavior of physical phenomena in terms of physical principles and is, generally speaking, not concerned with design of machinery and apparatus for manufacturing purposes. The engineer, on the other hand, is applying the principles of physics in the creation of machinery and apparatus for manufacturing or other useful purposes.

Throughout these differences noted between the engineer and a person in an adjoining field, one finds *ability to design* as a significant and distinctive element. The process of contriving a scheme, system, or concept of a device, together with a forecast of behavior thereof, which if built would be appropriate to the functional, economic, and safety requirements, is here meant as design. An engineer, for example, designs a bridge by first analyzing the conditions and functions to be satisfied; and secondly, by synthesizing from his knowledge of foundations, loadings, construction methods, mechanics, properties of materials, traffic requirements, economic considerations, financing, etc., the elements appropriate to this particular structure. It may now be stated that:

An engineer is characterized by his ability creatively to apply scientific principles to design or develop structures, machines, apparatus or manufacturing processes, or works utilizing them singly or in combination; or to construct or operate the same with full cognizance of their design, and of the limitations of behavior imposed by such design; or to forecast their behavior under specific operating conditions; all as respects a specific function, economics of operation, and safety to life or property.

The function of design is not limited to machines and structures. Process design may be defined briefly as the determination of the best process to accomplish a given end from the standpoint of economy, safety, raw materials, and available equipment. Courses in engineering application, therefore, which supplement those in applied science for full development of design qualifications are not limited to machine or structural design.

A curriculum which adequately trains a man to become an engineer must furnish the technical background, and should supply also the general training, for the process of analysis and synthesis essential to designing. There is the question, however, as to what may be considered an adequate training in the various essential features of the curriculum.

All engineering curricula may be divided into five sections as follows:

(1) basic science (mathematics, physics, chemistry, biology); (2) applied science (mechanics, thermodynamics, fluid mechanics, aerodynamics, geology, properties of engineering materials); (3) applied engineering courses (as for instance internal combustion engines, machine design, structures, industrial electronics, plant layout); (4) administrative and managerial (cost control, quality control, industrial organization, labor relations); and (5) general courses providing a general educational background.

Mathematics, physics, chemistry, and biology would be the fields of basic science from which the foundation of engineering training would be drawn, in amount appropriate to the particular branch of engineering.

New Bureau of Mines Personnel

The U. S. Bureau of Mines has released the names of those who head the various divisions and regional offices established by the reorganization of the Bureau, as reported in the September issue of Mining Engineering. They are as follows:

Director, James M. Boyd.

Assistant Director, Thomas H. Miller.

Assistant to the Director, Julian W. Feiss.

Chief counsel, John Hoffiund; chief metallurgist, Oliver C. Ralston; chief mining engineer, E. D. Gardner; office of mineral reports, Allan Sherman; chief of air and stream pollution research, Louis McCabe.

Region I, Alaska, Sinclair H. Lorain. Headquarters, Juneau. Region II, northwest, Stephen M. Shelton. Headquarters, Albany, Oregon. Region III, southwestern, Harold C. Miller. Headquarters, San Francisco. Region IV, Rocky Mountains, John H. East, Jr. Headquarters, Denver. Region V, North Central, Paul Zinner. Headquarters, Minneapolis. Region VI, south central, Dr. Clifford W. Seibel. Headquarters, Amarillo, Texas. Region VII, southeast, Hewitt Wilson. Headquarters, Norris, Tenn. Region VIII, northeastern, Harold P. Greenwald. Headquarters, Pittsburgh, Pa. Region IX, foreign minerals, Elmer W. Pehrson (acting).

Problems of a

Mine Mechanization Program

by C. Kremer Bain

To keep shovels working to capacity in the smaller and more scattered ore bodies necessitated the development of a mobile jumbo drill rig, utilizing the V-cut round drilled to a template.

The paper shows the steps taken, and the difficulties encountered in accomplishing an increase of 38 pct in tons per drill shift over the method previously used, with probability of more efficient operation to come.

UNDER present economic conditions the necessity of mechanizing the mines of our country has become a very important problem. More and more mines are looking toward increased or complete mechanization of their operations, so in the writer's opinion it is quite pertinent that some of the associated problems that accompany such a program be pointed out.

The following generalization on attempting a comparatively slight change in mining methods at the Southeast Missouri properties of the St. Joseph Lead Co. is an attempt to point out to anyone faced with a mechanization program that much time and effort can be saved, and many headaches avoided, if company policy can be established whereby everyone in the organization from the top down is desirous of improvement and is working towards the common objective. Any changes in mining methods, machinery, and so on, must entail coordination of all the other phases of mining as they all have to mesh as accurately as a smooth running chain of gears if satisfactory results are to be obtained.

C. KREMER BAIN, Member AIME, is Assistant to General Manager, St. Joseph Lead Co., Bonne Terre, Mo.

AIME San Francisco Meeting, February 1949.

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The purpose of this paper is to point out some of the problems which have to be surmounted when any mechanization program is attempted, rather than to go into the mechanical details and actual operating data of the equipment employed. After all, machines have to be adapted to best suit the local conditions, so no general formula can be given that would apply to other properties.

Although St. Joseph Lead Co. has been mechanically-minded for a number of years, as shown by the fact that over 110 million tons have been loaded mechanically since 1922, any further mechanization or changes always present a problem. When our ore bodies were of great height and area the breaking and loading were developed to a high degree of efficiency according to known standards. But in recent years, when our ore bodies became smaller and more scattered, the amount of tonnage we were able to break was not sufficient to supply our shovels with enough rock to keep them working to capacity. Our first step in attempting to rectify this situation was to develop a technique of moving the shovels to the broken ore in various stopes, which sometimes entailed a move each shift. A shovel is not producing when being moved, so within the last two years we have been endeavoring to devise some means of getting more rock broken in these smaller stopes and it was then that we became interested in jumbo drilling.

To illustrate the fact that mechanization is a process of a chain of developments rather than a stroke of genius can best be shown by a general summary of conditions and obstacles which were encountered in our present attempt to establish template drilling of a standardized round. Our first jumbo was more or less the conventional type with two machines mounted on a crossbar. This arrangement proved to be impractical to duplicate satisfactorily the regular routine of slabbing in a more or less radial pattern as carried out by column drilling. One machine was either too close to or too far from the face and there was another disadvantage in that one drillman was frequently delayed while waiting for his partner to bottom a hole. Our next step was to eliminate the crossbar so the machines could operate independently of each other, but we still encountered the same objections mentioned above except that one drillman was free to drill three holes instead of one in a vertical plane independently of his partner.

Time studies while trying to adapt the jumbo to the regular slabbing practice as accomplished by column drilling, revealed that twice as much time was spent in maneuvering the jumbo into position and aligning the holes as was consumed in drilling.

This page of Transactions AIME is a continuation from p. 128. The missing pages appeared in Mining Engineering.

We tried side swinging to eliminate some of this moving and were then confronted with a blasting problem. In an ordinary size heading it would take at least seventeen 3-hole settings to advance the stope from pillar to pillar. Blasting with fuse was impractical because in order to get the proper sequence of blasting, starting with a 6 ft fuse would require a minimum of 19½ ft of fuse for the last hole. With the 10-delay electric caps available on the market, we tried series blasting of five 3-hole settings at a time but inevitably the lead wires were broken by the concussion of the previous blast, making it necessary to go back into the smoke to reconnect, which is impractical in every way and is against good safety practice.

During this period we decided to get an accurate record of our standard drilling as practiced from columns. Each mine foreman selected a better than average driller and a better than average stope. These men were selected not only because of their ability as drillers but because they possessed a temperament that would not resent a close check upon their work. Each stope was surveyed daily and the advance plotted. The collars of each hole were located accurately along the hole alignment. This was checked the next shift by locating the points of the holes drilled the previous shift and measuring the bootlegs. From these data we gained some really valuable information as to the advance, burden, breaking angle, powder factor, percentage of holes broken and tons broken per foot of height.

Block models (scale 1 in. = 2 ft.) were made of each day's drilling and advance was colored for identification and the day-by-day progress studied closely. From these block models the present standard round was developed, applying the best results from column drilling as illustrated in fig. 1. We took sections marked 1 and 2 and placed them together, as shown in fig. 2, so we could incorporate more of the advantages of jumbo drilling than had heretofore been possible; namely,

1. Every man was free and independent of his partner for the entire shift's drilling.
2. Jumbo could be moved in on center line at the beginning of the shift and would not have to be moved after the original setting.
3. Permitted accurate spotting of holes as well as governing the depth of the holes to prevent over drilling, thereby minimizing the amount of bootleg.
4. Permitted use of the jumbo as a template to minimize lost motion and physical effort.

The solid block of ground remaining between the sections 1 and 2 left us with the problem of developing a dependable cut which would remove this "V" and still not deviate too far from our present drilling practice. The original efforts with this round met with only average results due to the inconsistency of pulling the cut. It took a lot of time and patience to develop the round to the present state, during which time we experimented with the size of the hole, kind and size of powder, and length of the powder column in relation to the

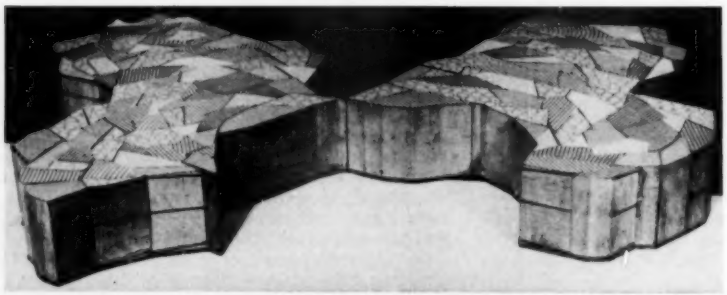

Fig. 1 — Block model showing conventional method of advancing a stope in Southeast Missouri room-and-pillar operations.

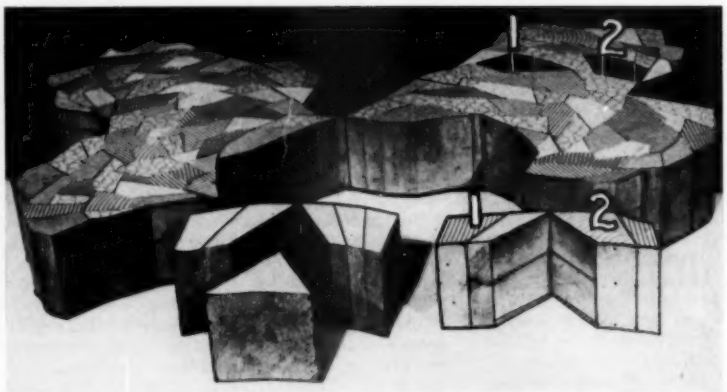

Fig. 2 — Same model as shown in Fig. 1 with sections 1 and 2 (which represent a part of four different shifts' drilling) removed and placed in foreground to compare with standard center-cut round.

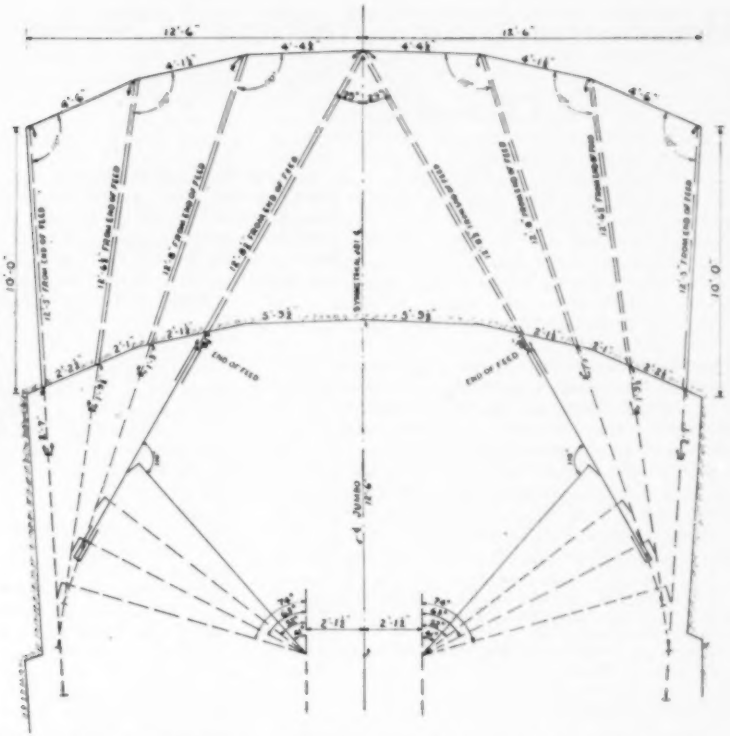

Fig. 3 — Plan view of standard round showing location of the center line of rotation of the jumbo arms and how it determines the depth of hole drilled to conform with the breaking angle of the rock. Eight setting round. 21.9 tons per foot of height.

length of the hole. During this time we had the cooperation and help of the technical representatives of powder companies as well as the miners themselves.

The present V-cut round, as shown in fig. 3, is the result to date of this study and experimentation. We think it has the advantage of using standard length of steel to control the depth of drilling. It will be noted from the diagram that all holes are bottomed at approximately the same required depth when the machine is at the same place on the feed.

In breaking the first 187,000 tons by Jumbo drilling, including all mechanical changes and the development of the V-cut round, we averaged only 43 pct of the tonnage which reasonably should be expected from a shift's drilling, and because of misbreaks, and so on, only 63 pct of the calculated tonnage was obtained from the round. This would have been most discouraging except for the fact that in spite of this poor showing our average tons per driller shift was 38 pct more than the average from column drilling during the period. At the present time our results are much more favorable and we have demonstrated that it is possible to get approximately 22 tons per foot of height in a face 25 ft wide, so we are quite confident that our efforts will confirm the feasibility of these methods.

During November 1948, some gratifying results were recorded. On one jumbo an average of 93 tons per driller shift was obtained for the 4-week period ending November 29, using the V-cut round. Another jumbo in a different mine obtained an average of 112 tons per driller shift for the last week in November with the V-cut round. This is the highest average per jumbo-driller shift obtained to date. It is interesting to note that the same crew drilling with the same jumbo in the same height of ground, but advancing the stope in the conventional way as employed in column drilling, averaged only 54 tons per driller shift during the month of October 1948. The change in method to the standard V-cut round, using the jumbo as a template, with somewhat better car service, was the only difference in conditions affecting these two results.

The development of the round drilled to a template has thus accomplished the original objective of providing sufficient broken ore in the smaller stopes to equal a shovel average that was obtained in the stopes of greater height and area. It has eliminated the excessive movement of shovel equipment but has necessitated closer supervision, rearrangement of haulage to provide adequate car service for the concentrated loading operations, and long range planning to coordinate loading and drilling so that the broken ore available for loading would not interfere with the operation of the jumbo.

In conclusion, we wish to emphasize the statement made earlier that changes in mining methods, machinery, and so on, must entail changes in and coordination with the other phases of mining.

Health and Safety Practices at Pioche

by S. S. Arentz

An organized safety program has reduced accidents at Pioche because effort is first devoted to arousing and maintaining interest in safety, followed by training in accident prevention, assigning responsibility for the detection and removal of safety hazards, and finally, by keeping score through an adequate system of records. A health program is intergrated with the safety program.

PLANNED health and safety programs have become an essential part of American industry because such programs lead to increased operating efficiency, improved labor relations, better public relations, and to substantial savings in compensation insurance. Those of you who have had the unpleasant duty of informing the wife or widow of one of your men of his serious injury or death while on the job, know that all the benefits of a successful safety program do not show on the balance sheet. These programs are of particular importance to the mining industry because mining's reputation as an unusually hazardous industry and the commonly isolated location of mining operations tend to focus attention on these problems.

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AIME San Francisco Meeting, February 1949.

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Description of Operations: Before proceeding with a discussion of our health and safety programs at Pioche, it may be proper to give a brief description of Pioche and of our operations there. Pioche is one of the early Nevada mining camps. It was founded shortly after the discovery of high grade silver ore in 1863 and mining has continued with more or less regularity to the present day. In an era of lawlessness, Pioche was notorious. The story persists that 75 men died with their boots on before one died a natural death, and old payroll records show that nearly as many gunmen were employed to stand off claim jumpers as there were miners working the mine. That was probably as close to a safety program as the times permitted. Pioche is situated in southeastern Nevada on the main highway between Ely and Las Vegas. The camp is on the flank of "Treasure Hill," near the original silver discovery, at an elevation of about 6000 ft. The present day population of about 2000 is primarily dependent upon the mines of the area, although Pioche also serves as the county seat of Lincoln County and as the center of the surrounding livestock industry. The camp is served by a branch of the Union Pacific Railroad and receives power from the generators at Hoover Dam.

The Pioche operations of the Combined Metals Reduction Co. were started in 1923 when the first complex lead-zinc ore was shipped to the company's mill at Bauer, Utah. The modern mill at Pioche was completed in 1941. The operations are medium sized in the nonferrous field, employing an average of 350 men in the mine, mill, and related works. The complex lead-zinc ore is mined from replacement deposits in a comparatively flat, extensively faulted, limestone horizon. Mining methods vary from stull-supported open stopes to filled square-set stopes. The thin bedded limestone and shale overlying the ore is allowed to cave as areas are mined out and caving frequently follows closely upon ore extraction. The relatively heavy ground and the numerous faults add to the problems of safe mining. The mine is well mechanized and the mill and surface plant are modern and well equipped. Labor is organized in a C.I.O. union and labor-management relations have been unusually harmonious.

During most of the period since 1923 a competent supervisory staff worked to reduce safety hazards but the primary responsibility for safety rested on the individual workman. Accidents happened and all too frequently they were regarded by all concerned as unavoidable.

In October 1939, the late Robert L. Dean became superintendent at Pioche. Most of his previous experience had been in the fields of iron and coal mining and from that experience he brought the concept that no accident is unavoidable. Many of the features of our present health and safety programs were initiated by Mr. Dean during his term as superintendent.

Health Program: Our health program centers in Dr. Q. E. Fortier and his new, well-equipped, and well-staffed, modern hospital in Pioche. The program starts with a thorough pre-employment physical examination and is followed by yearly re-examinations at the expense of the company. The Pioche Mutual Benefit Association, to which all Pioche mine operators and employees belong, pays benefits covering hospitalization and surgery expense incurred by employee members and their families. The Association is governed by a board of directors elected by its members. The mine operators of the district donated the original capital and pay the monthly dues of the employee members. The employees pay the dues covering members of their families. Though not strictly a part of the

health program, all employees are covered by group life insurance with the premium paid by the company. The union supplements the program for its members by the payment of compensation benefits to men losing time due to illness or nonindustrial accident.

The Pioche operations offer no unusual health hazard. Both the mine and mill are well ventilated with suitable dust-control equipment and practices. The underground temperature is roughly equivalent to the mean annual surface temperature and most workings are dry. Well-equipped change rooms, a clean boarding house, and above average bunkhouse facilities maintained by the company contribute to the health program.

Safety Program: Our safety program is the direct responsibility of our safety inspector, D. T. Sullivan, but all supervisors are responsible for the safety of the men under their supervision, and the union safety committee works closely with the management on matters relating to safety.

Inspection: Accident prevention starts with the detection and removal of safety hazards. The workmen, the supervisors, and the engineers are continually making informal safety inspections in the course of their regular duties and they automatically correct many hazards as they are noted. But some hazards are overlooked and within a short time become so familiar a part of the daily scene that they are completely disregarded. Other hazards are assumed to be the responsibility of someone else and are allowed to persist until they result in a serious accident. It is therefore essential that regular safety inspections be made and that responsibility for certain types of inspection be definitely assigned to individual members of the crew.

At Pioche we maintain the following safety inspections and the results of each inspection are entered on signed reports:

1. Daily inspection of all hoisting equipment including hoist ropes, safety and control devices, cages, skips, and hoisting compartments.

2. Semiweekly inspections of all active mine workings by safety inspector.

3. Semimonthly inspections of main shafts by maintenance crew.

4. Regular inspections by state mine inspector or his deputy.

5. Monthly inspection of all mine workings by employees safety group headed by the chairman of the union safety committee assisted by two experienced miners appointed by the management. The group is accompanied by the safety inspector, the mine foreman, or a shift boss.

6. Regular mechanical inspection covering condition of equipment, safety guards, and so on.

In addition to the above-listed, regular safety inspections there are numerous special inspections including the investigation of all lost-time accidents.

Safety inspections are of value only to the extent that they are followed by action designed to remove the hazards detected. At Pioche the reports on the various safety inspections are read by the department head concerned and by the plant superintendent and are then posted on the plant bulletin boards. In most cases the hazards noted are corrected immediately and a note is made on the report showing the action taken. Occasionally a recommendation in a report involves a material change in method, equipment, or policy, and such cases are considered by the plant and company management before action is taken. But in all cases some definite action is taken and noted on the posted reports before the reports are relegated to the files.

All lost-time accidents are investigated by an employee's committee appointed from the men working in the department in which the accident occurred, and by the safety inspector and the department head. The reports on the investigations show cause of accident, fix responsibility for the accident, and make recommendations for avoiding similar accidents. The reports are posted on the bulletin board and are distributed to the general manager, the department heads, and the local union. Our experience indicates that these reports have done much to reduce accidents and that the employees' committees take the job seriously and are fair in fixing responsibility.

Accident Prevention: Attention is given to the prevention of accidents through good housekeeping, the proper installation and use of equipment, and the use of protective equipment, warning signs, and other safety devices. The following details may be noted:

1. Machinery is equipped with safety guards and electrical installations are grounded.

2. Warning signs and signals are maintained. Scotchlite reflecting tape is widely used for signs underground and for indicating obstructions such as projecting chute lips. Patches of the tape on men's hats have prevented several possible haulage accidents.

3. All underground men wear hard-boiled hats, steel-toed boots, and battery lamps. Candles and matches are provided at convenient locations for use in testing air in workings off ventilation courses.

4. Entrances to abandoned workings are barricaded. Signs indicating nearest exit are posted at raise stations and drift junctions.

5. Well-equipped, well-maintained first aid stations are located on all levels underground and at central points on the surface.

6. Scaling bars and similar equipment are painted a bright yellow to aid in identification and as a reminder of their regular use.

Training: Training is an essential part of a safety program. We have made good use of the excellent training courses offered by the U.S. Bureau of Mines and have supplemented these courses with a training program conducted by our plant safety inspector. All Pioche supervisors and many key employees have completed the Bureau of Mines course in accident prevention and about 75 pct of our crew have completed the course in first aid to the injured. Many of our supervisors have instructor's certificates in first aid training. Attendance at accident prevention and first aid classes is voluntary and on the employee's own time.

We give regular training in mine rescue work to a picked crew, and annual classes in this subject are

conducted by a Bureau of Mines instructor. This training is given on company time.

The course conducted by the plant safety inspector covers accident prevention, plant safety rules, discussion of accident investigation and mine safety inspection reports, and demonstrations of safe methods of doing typical underground jobs. The classes are held on the main shaft stations underground just before a shift is hoisted to the surface after work. Classes are held biweekly for each crew and men attending class are delivered to the shaft station about 20 min before the end of the regular shift.

All employees receive a printed handbook containing the plant safety rules before reporting for work. In the past it was customary to penalize men for safety infractions with from one to ten days off without pay but under present conditions we find it better to caution men for minor infractions and to discharge them for repeated or flagrant violations. The rules were compiled and are occasionally revised by a committee representing both labor and management.

Accident Reports: Detailed records are necessary for a well planned safety program. Such records show progress made in safety, give a comparison of results achieved with those obtained at other mines or with past performance at the plant, and frequently reveal unsuspected hazards or dangerously careless employees.

At Pioche our records start with a foreman's report of accident on all injuries, even minor cuts and bruises. The reports are made in duplicate with one copy for the injured employee to give the doctor treating the injury, and the other for the department head and plant superintendent for their information before it is turned over to the safety inspector for his permanent files. A stub attached to the doctor's copy must be filled in showing the employee's release by the doctor before the injured employee may return to work. Failure to report injuries by the end of the shift in which they occur subjects an employee to immediate discharge. Prompt treatment of even minor cuts and scratches has eliminated infection following injury.

Other records include the various accident reports, safety inspection reports, reports on manshifts worked in the various departments and reports on days lost due to accident. From this information the safety inspector compiles records showing the location, cause, and extent of injury of each accident, the severity and frequency rates of lost-time accidents, and individual records covering the accident history of each employee. Monthly and yearly reports are prepared giving detailed information on the progress of our safety program with graphs showing comparisons with past periods at the plant and with national averages. The data are prepared to conform with the similar figures released by the U.S. Bureau of Mines.

Maintaining Interest: The problem of arousing and maintaining interest in safety is a most difficult one and its solution can assure a successful program. The training programs, the safety inspections, and accurate records showing progress made, all contribute to arousing and maintaining interest in safety, but in our experience it takes some additional incentive to keep interest alive over any extended period of time. We have found that if the interest of the supervisors can be aroused and maintained, the problem is more than half solved. One method we employ with success is a bonus system under which all supervisors completing a month without a lost-time accident among the men in their crew receive a cash bonus. The amount of the bonus varies depending upon the number of men supervised and the hazard rating of the department in which they are employed. A variation of this system has been a contest between the three mining operations and between the two mill and surface operations of the company. The contests run for periods of four months with a substantial cash bonus for the foreman and shift bosses at the mine and mill having the lowest lost-time frequency record. As the relative standings of the several operations are posted monthly, the contest serves to maintain a lively interest.

It is equally important and even more difficult to arouse and maintain the crew's interest in safety. Our methods have ranged from essay contests in the local schools to safety parties and dances. For the past few years we have had success with the following method:

All plant employees are divided into four groups with the men from the several departments distributed equally through each group. The men in any group which does not have a lost-time accident during the month receive free show tickets for themselves and their families. Needless to say, the man that knocks his group out of show tickets is unpopular with the group.

A second method regularly employed at Pioche is a bonus system in which the company contributes a substantial sum to the employees entertainment committee whenever the plant completes sixty days without a lost-time accident and a similar amount for each thirty days the accident-free period is extended.

A primary requisite to any of these methods is that there be no misunderstanding as to what constitutes a lost-time accident. We leave the decision up to the doctor on the understanding that the man does not return to work unless he can perform his regular duties without serious discomfort or danger to himself or others. We sometimes change the duties of injured men to allow them to draw wages instead of compensation, but all such cases are carried as lost-time accidents.

Attractive bulletin boards, posters, and signs all contribute to improved plant safety.

Achievements: Even though we have given a great deal of attention to the problem of preventing accidents, we are still far from our goal of an accident free year or years. We cannot fail to be impressed by the outstanding records achieved by mining operations placing high in the national safety competition. The methods outlined above have substantially improved our safety record. Experience has shown that there is no easy way to achieve a safe operation. Safety, like all other good things in life, is had only in proportion to the brains and effort devoted to it.

Faster Calculation of Plane Triangulation Systems by Calculating Machine and Semigraphical Methods

by Richard Hamburger

Calculating machines permit the use of the more rapid cotangent and semigraphic solutions of plane triangulation. The results of these methods are as accurate as those of other methods. Simple adjustment of the net, and ease of calculation encourage more complete field work. Errors in field work or in computation are easily localized.

(1) Introduction: The purpose of this paper is to call attention to certain methods which will shorten and simplify the calculation of plane triangulation. These methods, though not new,[1, 2] do not appear to have been properly publicized in the United States.

The cotangent (or tangent) solution of a triangle permits calculation of coordinates in a smooth series of operations of the calculating machine with a minimum of notation. The semigraphical solution permits the simultaneous solution of a number of intersections, at one station, without the added burden of solving additional triangles. This in turn encourages more numerous field checks and more complete office computation than would normally be undertaken.

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Tables I and II show the distinct advantage which the combined use of the cotangent and the semigraphical solutions hold over, what might well be called, the standard solution.

It will be shown that the positions for the entire net in fig. 1 may be calculated without computing any distances, thus saving much computation.

(2) Comparison of Cotangent and Standard Solutions: To compare the cotangent method with the standard method, the notation involved in the solution of triangle *1-X-4*, fig. 1, by both methods is shown. Fig. 2 shows the given information, that is, the coordinates and azimuths of Stations 1 and 4, and angles of the triangle or the azimuths from Stations 1 and 4 to Station *X*. (Azimuths are used here because the author feels that they are easier to handle, but bearings may be used by those who prefer them).

Cotangent Method

	Eastings	Northings		
Sta. 1	2,151.62	3,400.52	Cot 192° 33′ 30″	+4.489074
Sta. 4	6,181.59	1,125.70	Cot 142° 00′ 25″	−1.280262
			(See Sec 3)	+5.769336
Sta. X	2,651.608	5,645.00		

Standard Method

Sta. 1	2,151.62	3,400.52	Distance Sta. 1 to Sta. 4 by Pythagorean theorem on machine: 4,627.68
Sta. 4	6,181.59	1,125.70	
Diff.	4,029.97	2,274.82	

$$4{,}627.68 \times \frac{(\sin 22^\circ 33' 48'')\ 0.383704}{(\sin 50^\circ 33' 05'')\ 0.772195} = 2{,}299.50$$

$$4{,}627.68 \times \frac{(\sin 106^\circ 53' 07'')\ 0.956888}{(\sin 50^\circ 33' 05'')\ 0.772195} = 5{,}734.52$$

(sin 192° 33′ 30″) 0.217433 × 2,299.50 = +499.99

(cos 192° 33′ 30″) 0.976075 × 2,299.50 = +2,244.48

Sta. 1	2,151.62	3,400.52
	+499.99	+2,244.48
Sta. X	2,651.61	5,645.00

(sin 142° 00′ 25″) −0.615566 × 5,734.52 = −3,529.98

(cos 142° 00′ 25″) 0.788085 × 5,734.52 = +4,519.29

Sta. 4	6,181.59	1,125.70
	−3,529.98	+4,519.29
Sta. X	2,651.61	5,644.99

It will be seen from the above comparison that the advantages of the cotangent solution of a triangle are: (*a*) rapidity, (*b*) full use of the calculating machine, (*c*) the small number of

functions to look up, and (d) the small amount of notation involved. There is, however, one great disadvantage also apparent above. The cotangent solution, within itself, contains no checks. This disadvantage must be overcome by careful notation, and accurate extraction of functions.

(3) Solution of the Triangle by the Cotangent Method: The machine calculation of a triangle by

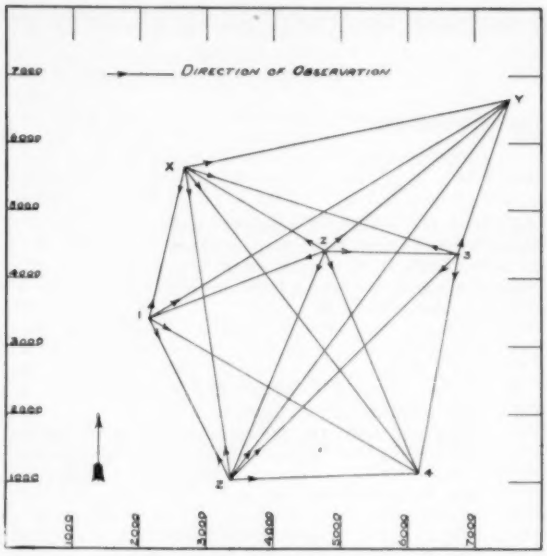

Fig. 1—Triangulation net.

the cotangent method is based on the following formulas whose derivation will be given in Sec. 8. The notation is that of fig. 3.

Given E_A, N_A, E_B, N_B, and azimuths α, and β, find E_C, N_C.

$$N_A + (E_B - E_A)\cot\alpha = N_D;\ E_B = E_D \qquad [1]$$

$$E_D + \left[\frac{(N_B - N_D)}{(\cot\alpha - \cot\beta)}\right] = E_E \quad = E_C \qquad [2]$$

$$N_D + (E_C - E_D)\cot\alpha = N_C \qquad [3]$$

The following notation[2] is complete and convenient, and embodies all of the writing involved in the solution of the equations.

Sta. A	E_A	N_A	$\cot\alpha$
Sta. B	E_B	N_B	$\cot\beta$
			$\cot\alpha - \cot\beta$
Sta. C	E_C	N_C	

The procedure of the calculation is as follows.[1]

1) *Set N_A on the product dials, E_A on the revolution dials, and cot α on the keyboard:* The decimal points must be so placed that they are consistent. The rule is that the sum of the decimal places on the revolution dials plus the sum of the decimal places on the keyboard equals the sum of the decimal places on the product dials. For example, if E_A is set on the revolution dials with the decimal point at three places, and cot α is set on the keyboard with the decimal point at six places, then N_A must be set on the product dials with the decimal point at nine places. These decimal point positions, once selected, are maintained throughout the calculation. It is advisable to set E_A as far to the left, and cot α as far to the right as is possible on the calculator. The exact position will vary with the capacity of the machine at hand. The selection of the decimal point positions should permit the value of the last significant number of interest to be no longer affected by further operations of the calculating machine. By operating the calculator is meant the turning of the crank or other motivating device in either a positive or a negative direction.

2) *Operate the calculator so as to change E_A on the revolution dials to read E_B.*

3) *Clear the keyboard only and set (cot α — cot β) on the keyboard.*

4) *Operate the calculator so that the product dials read N_B. Read E_C on the revolution dials and write it down.* During this step carry E_C one or more places beyond the last significant number desired, and use this value in calculating N_C in step 7. In performing steps 4, 5, and 7 on calculating machines having no carryover mechanism on the revolution dials, care must be exercised so as not to allow numerals of opposite sign to appear on the revolution dials. Thus if black (positive) numerals are being used, the machine must be operated so as to maintain these numerals black (positive) throughout the calculation. Since this step is a division it should start at the left hand part of the revolution dials and progress to the right (that is, the carriage moves to the left). To maintain numerals of the same sign, on calculating machines with no carryover mechanism on the revolution dials, may require some juggling of the carriage between two adjacent columns. Thus if the machine is being operated in a subtractive manner and it is found that, say, 30 (black) is too large a number on the revolution dials and that 20 (black) is too small a number on the revolution dials to allow the numerals on the product dials to approach the value of N_B, make the numerals on the revolution dials read 29 (black) and proceed from there to find that number, between 30 (black) and 20 (black), which best satisfies the condition sought. A similar operation may be performed if the machine is being operated in an additive manner and

Table I. Comparison of Methods in Single Solution

To Solve Triangle and Check	Standard Solution	Cotangent and Semigraphical Solution
Number of functions looked up	7	4
Operations of machine	11	7
Number of series of operations	5	3
Sum = Chance of error and speed of operation	23	14

The addition of one extra observation brings the possible answers to three instead of one, and gives table II.

it is found, for instance, that 59 (black) is too small and that 69 (black) is too large. Make the numerals on the revolution dials read 60 (black) and proceed with the step. At all times the black (positive) numerals must be used. The computer should guard against going through the red (negative) numerals and ending again with black (positive) numerals. If the calculation is started with the red (negative) numerals, then these must be maintained in a similar manner, remembering that the zero is usually common to both series.

5) *Without changing any setting, operate the calculator so that the revolution dials read* E_B.

6) *Clear the keyboard only, and set cot* α *on the keyboard.*

7) *Operate the calculator so that the revolution dials read* E_C. *Read* N_C *on the product dials.*

The following rules must be adhered to.

1) The signs of cot α and of (cot α — cot β) must be the same.

2) If the sign of cot α is positive the original setting of E_A on the revolution dials uses the positive numerals. If the sign of cot α is negative the original setting of E_A on the revolution dials uses the negative numerals. On calculating machines having carryover mechanism on the revolution dials the sign of the numerals on this set of dials is controlled by a key. On calculating machines which do not have the carryover mechanism on the revolution dials there are two series of numbers, one positive and one negative, on the revolution dials. These two series are usually of different colors, such as black for the positive series and red for the negative series. The positive series is the one normally used in multiplication and the negative series the one normally used in division. As noted in step 4 of the procedure the use of either the positive or negative setting of the revolution dials must be maintained throughout the calculation.

3) The signs of E_A and E_B, and the signs of N_A and N_B must be the same. As it is convenient to have all of the calculations in the northeast quadrant, a suitable constant may be added to the coordinates when convenient.

Cotangents near 0°, and those near 180° are such large numbers that their use makes the results unreliable. In such cases the tangents of the azimuths are used. Use the same notation and the same steps, but read tangent for cotangent, easting for northing, and northing for easting.

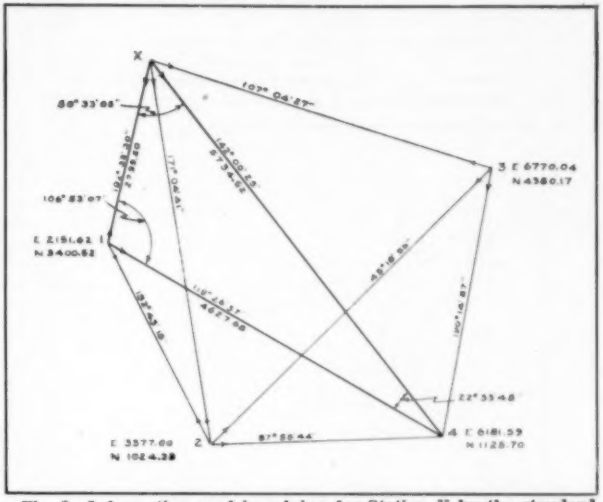

Fig. 2—Information used in solving for Station *X* by the standard method, and by the cotangent method.

(4) Semigraphic Computation: Semigraphic computations,[5, 6, 8] because of their simplicity, are one of the greatest time savers in the computation

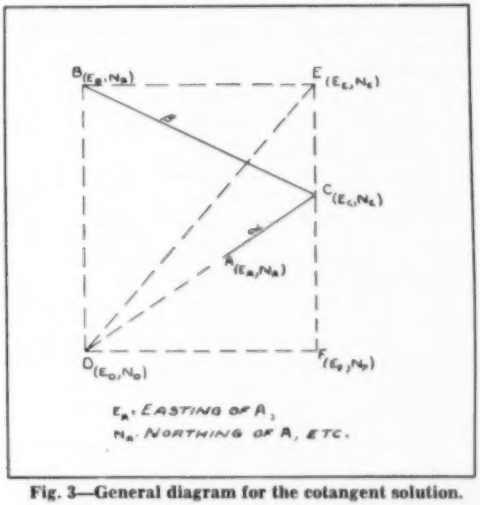

Fig. 3—General diagram for the cotangent solution.

Given the positions of *A*, *B*, and *C*, and the azimuths α and β from *A*, and *B* to *C*. Construction: (1) Extend α to intersect E_B. Call this position *D*. (2) Call the intersection of E_C and N_B, *E*. (3) Complete the rectangle. (4) Connect *D* to *E*.

Table II. Comparison of Methods in Multiple Solution

To Solve Triangles and Check	Standard Solution	Cotangent and Semigraphical Solution
Number of functions looked up	21	3
Operations of machine	33	9
Number of series of operations	15	3
Sum = Chance of error and speed of operation	69	15

of plane triangulation. The method consists in calculating the intersection of azimuths, from known stations, with coordinate axes of a position which is approximately correct for the unknown point. Having computed these intersections the information is plotted on graph paper and the intersection of the azimuths (if only two) is scaled. If there are more than two azimuths plotted, an error figure will result and a point must then be selected, which will best fit the given information.

The intersection of the azimuths at Station X from Stations 1 and 4 has been computed in the example showing the computation by the cotangent method. Referring to fig. 1 it will be seen that

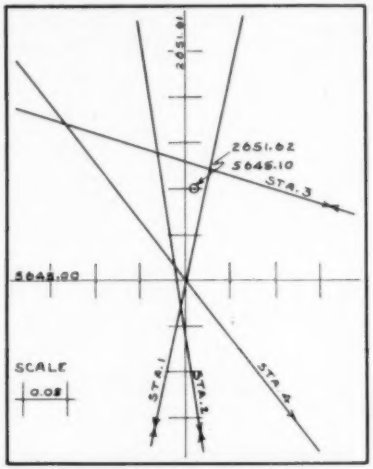

Fig. 4—Error figure for Station X.

there are also observations including Stations 2 and 3. Using the position previously obtained as an approximate position for Station X the following notation illustrates the remaining calculation; the azimuths from Stations 2 and 3 are given.

	Sta. X	2,651.61	5,645.00		
	Sta. 2	3,377.00	1,024.28	171° 04′ 41″	—0.156988
Diff.	= —0.01	2,651.60	4,620.72		
	Sta. 3	6,770.04	4,380.17	107° 04′ 27″	+0.307147
		4,118.43	5,645.13	+0.13 = Diff.	

The results are then plotted on graph paper to a suitable scale, fig. 4. It is evident from fig. 4 that the above computation and plotting is equivalent to the solving of five additional triangles. The results are as accurate as any numerical solution, and require far less labor.

The advantages of the semigraphical solution are: (a) rapidity of computation without loss of accuracy, (b) the ease with which errors may be located (either errors in computation, or errors in field work). To illustrate point (b) an error will be purposely introduced into two of the illustrative problems.

The semigraphic solution is based on the following formulas[7, 8] whose derivation will be given in Sec. 8. The notation is that of fig. 5. Given E_A, N_A and azimuth α find the intersection of α with coordinates E_B, N_B.

$$E_B = E_A + (N_B - N_A) \tan \alpha \quad [4]$$

$$N_B = N_A + (E_B - E_A) \cot \alpha \quad [5]$$

Only one of these need be calculated since the line may then be plotted with a protractor. When α is close to 0°, 90°, 180°, 270°, and 360°, one of the axis intersections may lie quite far from the approximate position chosen. It is therefore convenient to work with the equation which will give the closest axis intersection.

The following[9] rule will produce this result.*

* This rule can be used only with natural function tables arranged so that reading from the top forwards are values from 0° to 44°, and from the bottom backwards values from 45° to 89°. Most function tables are arranged in this manner.

1) For the function of α always use the value of the column headed tangent at the top of the page. This value will always be 1.000000 or less.

2) Find the greatest difference in coordinates between A and B; either $(E_B - E_A)$ or $(N_B - N_A)$.

3) Multiply this difference by the function of α and add algebraically to that part of the coordinate of A which was not used in step 2. The result will be the closest intersection of α with one of the coordinate axes of B. The multiplication and addition should be combined on the calculator so that only the final answer need be recorded.

The above rules, like all rules, should at times be violated. Thus this calculation is a convenient check for errors introduced into the computation by the cotangent method, but it would be dangerous to re-use a function.

(5) The Error Figure: When more than two azimuths are observed to a point there will in all likelihood be an error figure (see fig. 4). This error figure consists of the area inclosed by the azimuths, their points of intersection, and the area immedi-

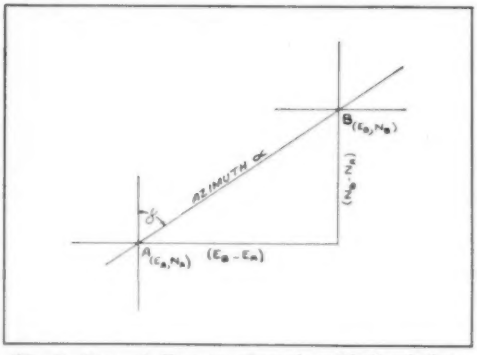

Fig. 5—General diagram of semigraphical solution.

ately adjacent thereto. The error figure will be a triangle in the case of three observed azimuths, and some more complicated figure when there are more than three observations.

The problem of the error figure consists in finding that point which most nearly satisfies all of the observations. This point, when chosen, will be the final coordinates of the station. The following rules are used in selecting the point:[10, 11]

1) The point lies closest (perpendicularly) to the azimuth from the nearest station, and furthest from the azimuth from the furthest station.

2) The point lies in such a position that, re-

gardless of sign, the change in azimuth caused by this selection is roughly the same for all observations.

3) Azimuths observed in both directions carry more weight than those observed in only one direction. This is obviously not true if only one azimuth has been observed in both directions.

Graphs and tables[15, 16] may be constructed showing the relationships in rule 2, and judgment must be used in applying rule 3. It will be found that, after slight practice, the position may be selected by eye with nearly the same accuracy.

The error figure permits the detection of mistakes. An azimuth which plots out of sympathy should be checked for mistakes in computation and field work. If no mistakes are found in either of these, and the position of the fixed station is not in doubt, then this azimuth should exert its proportional strength in the error figure.

(6) Direction and Azimuth Adjustments: In a triangulation net which consists of figures more complicated than chains of simple triangles, some form of adjustment must be made so that the results will be the most probable positions for the observations made. For precise work an adjustment by the method of least squares[14] would be justified. For most work a system which approximates this, and which is easy to handle, will suffice. Such a system is found in the direction and azimuth adjustment,[15, 16] (see table III), used in connection with the error figure. This notation consolidates all of the angular information from the beginning of the computation to the final (adjusted) azimuths. This notation permits the rapid extraction of data from separate observation stations pertinent to one problem. The great advantage of this method of handling the observations in conjunction with the cotangent solution is that it permits easy adjustment of the azimuth while eliminating the necessity of the intervisible base line. Thus the observation stations may be chosen for their convenience. As long as all stations used in determining azimuths are in sympathy this introduces no additional error, and is, in many instances, a great help to the observer to whom it allows a greater flexibility.

The form consists of eight vertical columns, and a separate block for each station occupied. Column 1 shows the Station Occupied, followed by the Stations Observed from it. Column 2 shows the Observed Directions. Column 3 gives the Approximate Azimuths from the station occupied to the stations observed. It is convenient to make these values less than the correct values so that the signs in columns 5 and 7 may be positive. Column 4 shows the "Forward Azimuths." These are the "Back Azimuths" values transferred where pertinent. They only appear when the station occupied is unknown, and one or more directions have been observed in both directions. Column 5 contains the Provisional Corrections. This is the difference between column 4 and column 3. Column 6 gives the "Back Azimuths." This value is the sum of the value in column 3 and the mean of the Final Corrections in column 7, or the sum of the value in column 3 and the mean of the Provisional Corrections in column 5, depending on the case. Column 7 contains the Final Correction. This value is the difference between column 8 and column 3. The consistency, not the absolute values, of the figures in column 7 is a measure of the accuracy of the survey. Column 8 shows the Final Azimuths. These are calculated from the final coordinates.

Table III. Directions and Azimuths

(1)	(2) Observed Direction	(3) Approximate Azimuth	(4) Forward Azimuth	(5) Prov. Corr.	(6) Back Azimuth	(7) Final Corr.	(8) Final Azimuth
	° ′ ″	° ′ ″	′ ″	″	′ ″	″	° ′ ″
Sta. 1							
X	00 00 00	12 33 17			33 29	+12	12 33 29
Y	46 17 11	58 50 28			50 40	+12	58 50 40
4	106 53 10	119 26 27				+10	119 26 37
2	140 09 44	152 43 01				+14	152 43 15
						+12	
Sta 2							
1	00 00 00	332 42 50				+25	332 43 15
X	18 21 23	351 04 13			04 40	+29	351 04 42
Y	63 50 06	38 32 56			33 24	—9′38″	38 23 18
3	72 35 30	45 18 29				+26	45 18 55
4	115 12 24	87 55 14				+30	87 55 44
						~~+27~~ *	
						+38	
Sta. 3							
4	00 00 00	190 14 17				+40	190 14 57
2	35 04 18	225 18 35				+20	225 18 55
X	96 49 41	287 03 58			04 28	+28	287 04 26
Y	188 03 56	16 16 13			16 42	+28	16 16 41
						~~+30~~ *	
						+29	
Sta. X							
Y	00 00 00	78 22 46			22 38	+12	78 22 57
3	28 41 28	107 04 14	04 28	+14	04 26	+12	107 04 26
4	63 37 27	142 00 13			00 25	+15	142 00 28
2	92 41 44	171 04 30	04 40	+10	04 42	+13	171 04 43
1	114 10 32	192 33 18	33 29	+11	33 30	+11	192 33 29
				+12		+12	
Sta. Z							
1	00 00 00	249 05 15				+03	249 05 18
X	51 16 26	300 21 41				—04	300 21 37
Y	161 40 01	50 45 16				+00	50 45 16
3	201 43 09	90 48 15				—10″12″	90 38 03
4	267 39 21	156 44 36				—04	156 44 32
2	313 22 22	202 27 37				—07	202 27 30

* Crossed out.

Table IV. List of Coordinates

Sta. No.	E	N
1	2,151.62	3,400.52
2	3,377.00	1,024.28
3	6,770.04	4,380.17
4	6,181.59	1,125.70
X*	2,651.62	5,645.10
Y*	7,519.88	6,645.95
Z*	4,773.33	4,402.27

* These coordinates calculated, the others are given.

The following is an example of the use of the direction and azimuth adjustment. To be determined: the azimuths to be used in fixing the position of Station *X* in the net. The coordinates of Stations 1, 2, 3, and 4 are given. The Observed Di-

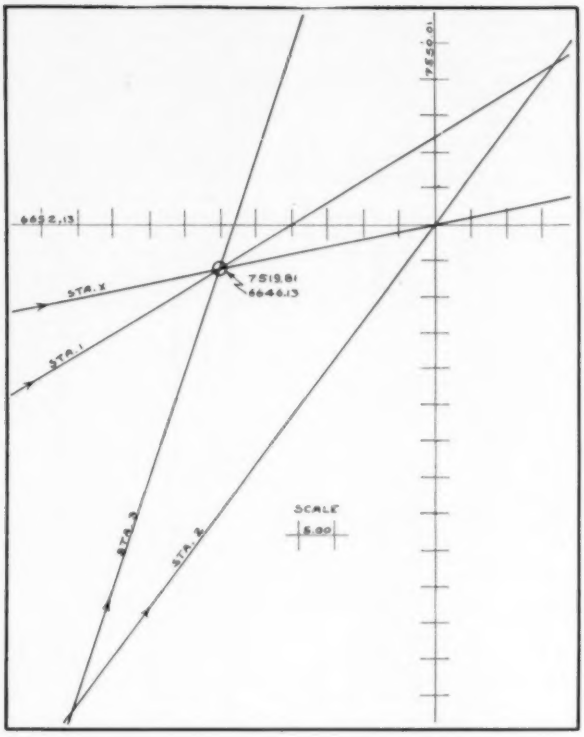

Fig. 6—First error figure for Station *Y*.

Note the obviousness of the errors in the answers depending on the azimuth from Station 2.

rections in column 2 are given. The coordinates of Stations *X*, Y, and *Z* are unknown.

Step 1: Compute the azimuths between Stations 1, 2, 3, and 4 for those directions which have been observed, and enter these values in column 8.

Step 2: Orient in column 3, to a value near those in column 8, the Observed Directions of column 2. Thus under the observations from Station 1, the direction to Station 4 has been oriented 10″ lower than the value in column 8, and the other directions have been oriented accordingly.

Step 3: Subtract the values in column 3 from those in column 8 and enter the results in column 7. Mean the values in column 7 and enter the results at the bottom of the blocks. Thus under Station 2 the Final Corrections to Stations 1, 3, and 4 are +25″, +26″, and +30″, respectively, and the mean is +27″.

Step 4: Add the means from column 7 to the Approximate Azimuths of Station *X* as found in column 3 in the blocks under Stations 1, 2, and 3, and enter these values in column 6. The degrees of column 3 are understood to apply to columns 4 and 6. The values in column 6 now give the most probable values for the azimuths, from Stations 1, 2, and 3 to Station *X*, without consideration of the Observed Directions at Station *X*. The next steps combine these results with the Observed Directions at Station *X*.

Step 5: Abstract those values from column 6 which appear opposite Station *X* under Stations 1, 2, and 3, and enter them in column 4 under Station *X* opposite the appropriate station. Thus from column 6 under Station 3, opposite Station *X*, abstract 04′ 28″ and enter in column 4 under Station *X* and opposite Station 3.

Step 6: Orient the Observed Directions under Station *X* to be near the values in column 4 and enter in column 3. The degrees will be 180° from those in the previous blocks.

Step 7: Subtract the values in column 3 from those in column 4 and enter in column 5.

Step 8: Mean column 5 and add the mean to the values in column 3, entering the results in column 6. This is done for the Approximate Azimuths of all stations having coordinates known at this moment. Thus Station 4 is brought into the picture, at this point, as an azimuth observed in one direction only. In computing the position of Station *X* the mean values between columns 4 and 6 are used. When the coordinates of Station *X* are fixed, the Final Azimuths to Stations 1, 2, 3, and 4 are computed and entered; the Final Correction is calculated and meaned, or remeaned as in the blocks under Stations 1, 2, and 3, and the sheet is ready for the abstraction of the information needed to compute Station Y.

(7) Applications: There are in general three problems which arise in the computation of triangulation. They are: (1) the unknown station is

occupied and two or more of the azimuths are observed in both directions; (2) the unknown station is not occupied but is observed from two or more stations; (3) the unknown station is occupied but is not observed both ways along a common azimuth. The first problem is, of course, the one for which to strive. The closed triangle and the closed quadrilateral are two special versions of this problem. The second problem is that of the point determined by intersection. The third problem is the three point problem. A point may be determined by any of these problems or by a combination of them.

Problem No. 1, the closed figure: the solution for Station *X* illustrates this case. The mean of the Forward and Backward Azimuths from the Direction and Azimuth sheet (table III) are used. There are four azimuths to Station *X* and therefore six possible intersections. The first intersection, that of Station 1 and Station 4, is computed by the cotangent method. Since the signs of the azimuths are opposite it makes no difference which is taken as α, since the sign of (cot α — cot β) will be the same. Computing the intersections of the azimuths to Stations 2 and 3 (with the coordinate axes determined by this first intersection) and plotting the results determines the five other intersections (fig. 4). The error figure is then evaluated.

Sta. 1	2,151.62	3,400.52	192° 33′ 30″	+4.489074
Sta. 4	6,181.59	1,125.70	142° 00′ 25″	—1.280262
				+5.769336
Sta. *X*	2,651.608	5,645.00		
Sta. 2	3,377.00	1,024.28	171° 04′ 41″	—0.156988
—0.01	2,651.60	4,620.72		
Sta. 3	6,770.04	4,380.17	107° 04′ 27″	+0.307147
	4,118.43	5,645.13	+0.13	

Problem No. 2, the intersection: the solution for Station *Y* is an illustration of this case. In this illustration the azimuth from Station 2 has an error of nine minutes. A similar result would occur if the coordinates of Station 2 were in error, or if a like error were made in computation. There are six possible answers to this problem; three of which are wrong. An examination of the numerical results of the first set of calculations will show that there is an error. By plotting the information (fig. 6) the wrong answers are immediately apparent. An approximately correct coordinate is scaled, and the intersections are recomputed without the need of looking up any new functions. The results are plotted (fig. 7). The calculations involving Station 2 are of course checked for errors in computation.

In calculating the first intersection the azimuth from Station 2 must be taken as α so that its sign and that of (cot α — cot β) will be the same.

Sta. 2	3,377.00	1,024.28	36° 33′ 24″	+1.348631
Sta. *X*	2,651.62	5,645.10	78° 22′ 58″	+0.205584
				+1.143047
Sta. *Y*	7,550.010	6,652.13		
Sta. 1	2,151.62	3,400.52	58° 50′ 40″	+0.604562
	5,398.39	6,664.18	+12.05	
Sta. 3	6,770.04	4,380.17	18° 18′ 42″	+0.330944
—28.08	7,521.93	2,271.96		

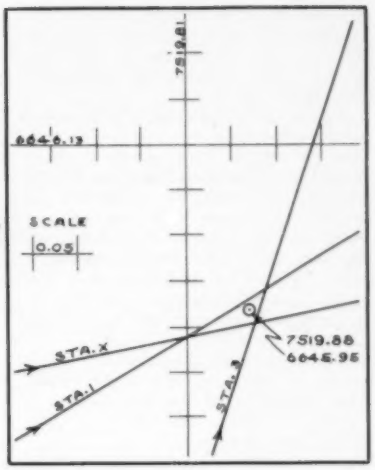

Fig. 7—Final error figure for Station *Y*.

Check Computation:

Sta. 2	3,377.00	1,024.28	36° 33′ 24″	+0.741493
±0.00	7,550.01	5,627.85		
Sta. *X*	2,651.62	5,645.10	78° 22′ 58″	+4.864197
±0.00	7,550.01	1,007.03		

Recompute:

Sta. *Y*	7,519.81	6,646.13	(Scaled from fig. 6)	
Sta. *X*	2,651.62	5,645.10	78° 22′ 58″	+0.205584
	4,868.19	6,645.92	—0.21	
Sta. 1	2,151.62	3,400.52	58° 50′ 40″	+0.604562
	5,368.19	6,645.92	—0.21	
Sta. 3	6,770.04	4,380.17	18° 18′ 42″	+0.330944
+0.14	7,519.95	2,265.96		

Problem No. 3, the three point problem: the solution for Station *Z* is an example of this case. There are numerous solutions of the three point problem but regardless of the solution used the essential part of the solution is the determination of the proper orientation of the observed directions. One solution is the auxiliary point method or Collin's solution;[17] the usual solution is the auxiliary angle method or Chauvenet's solution.[18] Of the two methods Breed and Hosmer state that the second ". . . is shorter and has a better arrangement for logarithmic computation".[19] While it is correct that the arrangement of the auxiliary angle solution is better for logarithmic computation, it is not the shorter solution if the cotangent method is used in solving the auxiliary point method. In the auxiliary angle solution there are six logarithms and two antilogarithms to look up to find the angles used in orienting the directions. In the auxiliary point solution there are two functions and one antifunction to look up. While in the auxiliary angle method the distances of two sides must be computed as a part of the preparation of the problem, in the auxiliary point solution no distances need be computed.

Having determined the proper orientation, solution of the triangle by the cotangent method gives the first approximate coordinate for Station *Z*, and solution for the intersection of the third azimuth (it should be 0.00 units) gives a check of the computation. It will be found that for certain azimuths

the tabular differences of the tangent and the cotangent will introduce a variation up to 0.01 units in the intercept. Since this is well within the limits of the error of observation the calculation may be considered checked.

Any three angles and any three coordinates will give a self consistent answer in a three point problem. A fourth observation is thus needed as a check. This fourth observation will give three more possible answers since neither the positions nor the angles are perfect. All four answers are equally valid, and the most nearly correct position must be the one that most nearly satisfies them all. To solve for all of these answers, numerically, by either method discussed runs into a lot of work. A semigraphical solution is therefore preferable. Lehmann's solution,[2, 3, 4, 5] so commonly used in plane table work, is very satisfactory. The intercepts of the additional azimuths are computed and plotted (fig. 9). The azimuths are then increased (or decreased) by an arbitrary amount (five seconds in the example) and these results are plotted (the dashed lines). A line (V-line) connecting similar vertices of these two orientations is then the line of all possible intersections of these two lines in all possible orientations. The V-line is actually a segment of a circle but for the short interval considered a straight line is used. Draw all of the V-lines and the points of intersection of three V-lines are possible solutions of the three point problem. In each case the three intersecting V-lines must have the same sense.

In the calculation of the auxiliary point Q, in problem Z, it will be found that the easting value is a negative sum. An addition of 20,000 units to the easting and northing of Stations 1, 4, and Y will place the computation completely in the first quadrant. In the solution of the triangle for point Q the azimuth from Station 1 must be taken as α since both azimuths have the same sign. Since this sign is negative the revolution dials must be operated with a negative setting.*

Sta. 1		00° 00′ 00″	(Abstracted from table III)		
Sta. Y		161° 40′ 01″			
Sta. 4		267° 39′ 21″			
Z4 — ZY	=	105° 59′ 20″	= γ	(See fig. 8)	
Z1 — Z4	=	92° 20′ 39″	= δ		
Azimuth 1Y	=	58° 50′ 40″			
Azimuth YQ	=	58° 50′ 40″ +	180° + (180° — 92° 20′ 39″)	=	326° 30′ 01″
Azimuth IQ	=	58° 50′ 40″ —	(180° — 105° 59′ 20″)	=	344° 50′ 00″

Sta. 1	22,151.62	23,400.52	344° 50′ 00″	—3.689093
Sta. Y	27,519.88	26,645.95	326° 30′ 01″	—1.510851
				—2.178242
Q	16,938.209	42,633.28		
Sta. 4	26,181.59	21,125.70		
Q	16,938.21	42,633.28	9,243.38/—21,507.58	= —0.429773
	+9,243.38	—21,507.58	tan (—0.429773)	= 156° 44′ 36″
Sta. Y	7,519.88	6,645.95	50° 45′ 16″	+0.816905
Sta. 4	6,181.59	1,125.70	156° 44′ 36″	—2.326817
				+3.143722
Sta. Z	4,773.388	4,402.33		
			(20″)	+(0.382085)
Sta. 1	2,151.62	3,400.52	249° 05′ 15″	+0.382113
	2,651.77	4,402.33 ±0.00		
		(4,402.26)(—0.07)		
			(46″)	—(0.585824)
Sta. X	2,651.62	5,645.10	300° 21′ 41″	—0.585791
	2,121.77	4,402.19 —0.14		
		(4,402.12)(—0.21)		
Sta. 3	6,770.04	4,380.17	90° 48′ 15″	+0.014036
	1,996.65	4,408.19 +5.86		
			(42″)	+(0.413430)
Sta. 2	3,377.00	1,024.28	202° 27′ 37″	+0.413402
+0.10	4,773.49	3,378.05		
(+0.20)	(4,773.59)			
Sta. Y	7,519.88	6,645.95	50° 45′(21″)	—(0.816865)
	2,746.49	(4,402.44)(+0.11)		
Sta. 4	6,181.59	1,125.70	156° 44′(41″)	—(0.429743)
(+0.09)	(4,773.48)	3,276.63		

* Calculating machines not having the carry over mechanisms on the revolution dials are usually equipped with a set of red numerals for negative operations. E_1 is then set in the red numerals and these are maintained on the revolution dials during the calculation.

It is seen from the calculation that the azimuth from Station 3 does not check the computation. A quick recomputation of this azimuth shows that there is no error in the calculation. Therefore the error must be either in the coordinates of Station 3 or in the observation. Since we know from the previous work that the coordinates of Station 3 are consistent with the net, the observation must be in error. It is therefore disregarded. This leaves five observations, giving a total of ten possible answers. It will be seen that the V-lines giving the solution using Stations 1, 2, and 4 approach parallelism, an indication that Stations 1, 2, and 4 determine a circle passing close to Station Z. This approaches the indeterminate case, so in evaluating the error figure this point is disregarded. (See table below.)

From Station Z six observations were made. There are then, to start with, twenty possible answers. Since the observation to Station 3 is in error, there are ten wrong answers out of twenty. There is a one to one chance of picking a wrong combination for the first try; there is a nine to ten chance of picking a wrong answer on the second try if the problem were solved entirely by numerical calculation. If the second try is a right answer the third try must be a correct one if there is to be a check, and the chances of this occurring are again one to one if the first try were wrong, and the second try were right. Lehmann's semigraphical

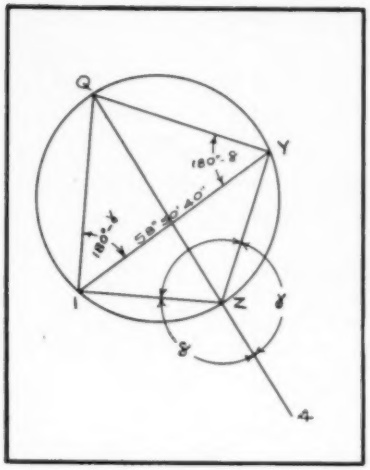

Fig. 8—Collins' solution of the three point problem.

solution avoids all of this calculation. If the first calculation had used the observations to Stations 3, 4, and *Y*, none of the extra intercepts would check (fig. 10). The rotated azimuths are calculated and plotted, and the V-lines are drawn. The vast majority of the intersections fall in the heavily circled area. A position is scaled here, and a recomputation will give the answer in fig. 9. An inspection of the V-lines passing through the other intersections show that the observation to Station 3 is involved in all of them. The error, whether of computation, of coordinate, or of observation is thus quickly localized.

(8) Derivations: Cotangent Formulas[24] (fig. 3): Given Station A (E_A, N_A), Station B (E_B, N_B), and azimuths α and β from Stations A and B to Station C.

Find Station C (E_C, N_C).

(1) $$\frac{N_C - N_A}{E_C - E_A} = \cot \alpha, \quad \frac{N_C - N_B}{E_C - E_B} = \cot \beta$$

(2) $$N_C = N_A + (E_C - E_A) \cot \alpha = N_B + (E_C - E_B) \cot \beta$$

(3) $$N_A + E_C \cot \alpha - E_A \cot \alpha = N_B + E_C \cot \beta - E_B \cot \beta$$

(4) $$E_C (\cot \alpha - \cot \beta) = (N_B - N_A) - E_B \cot \beta + E_A \cot \alpha$$

Add and subtract $E_B \cot \alpha$ to the right hand member of Eq 4.

(5) $$E_C (\cot \alpha - \cot \beta) = (N_B - N_A) - E_B \cot \beta + E_A \cot \alpha + E_B \cot \alpha - E_B \cot \alpha$$

(6) $$E_C (\cot \alpha - \cot \beta) = E_B (\cot \alpha - \cot \beta) + (N_B - N_A) - (E_B - E_A) \cot \alpha$$

(7) $$E_C = E_B + \left[\frac{(N_B - N_A) - (E_B - E_A) \cot \alpha}{(\cot \alpha - \cot \beta)}\right]$$

(8) $$N_B = N_A + (E_B - E_A) \cot \alpha$$

(9) $$N_A = N_B - (E_B - E_A) \cot \alpha$$

(10) $$E_C = E_B + \left[\frac{(N_B - [N_D - (E_B - E_A) \cot \alpha]) - (E_B - E_A) \cot \alpha}{(\cot \alpha - \cot \beta)}\right]$$

(11) $$E_C = E_B + \left[\frac{N_B - N_D + E_B \cot \alpha - E_A \cot \alpha - E_B \cot \alpha + E_A \cot \alpha}{(\cot \alpha - \cot \beta)}\right]$$

(12) $$E_C = E_B + \left[\frac{N_B - N_D}{(\cot \alpha - \cot \beta)}\right]; E_C = E_B; E_B = E_D$$

Add and subtract $E_B \cot \alpha$ to the right hand member of Eq. 2.

(13) $$N_C = N_A + (E_C - E_A) \cot \alpha + E_B \cot \alpha - E_B \cot \alpha$$

(14) $$N_C = N_A + (E_B - E_A) \cot \alpha + (E_C - E_B) \cot \alpha$$

Substitute Eq 8 in Eq 14.

(15) $$N_C = N_B + (E_C - E_B) \cot \alpha; E_B = E_D$$

Semigraphic Intersection Formulas[25] (fig. 5).

Given Station A (E_A, N_A) and Station B (E_B, N_B) and the azimuth from A to B.

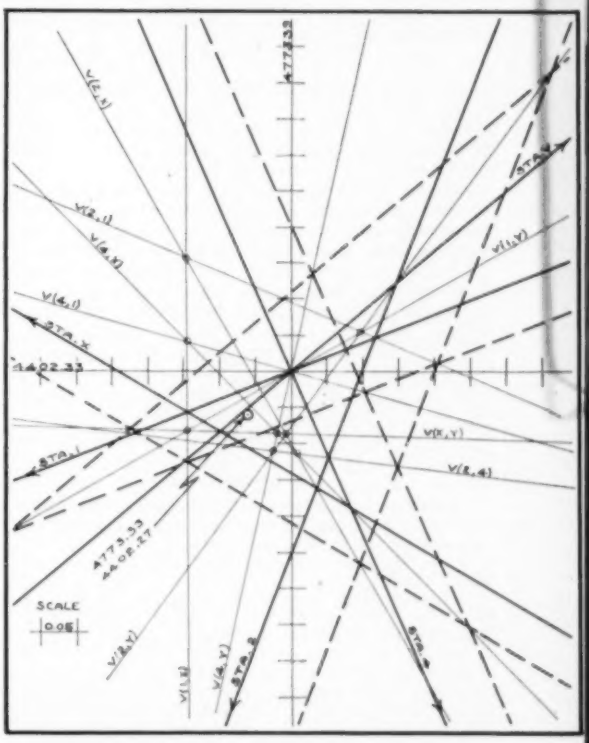

Fig. 9—Error figure for Station Z.

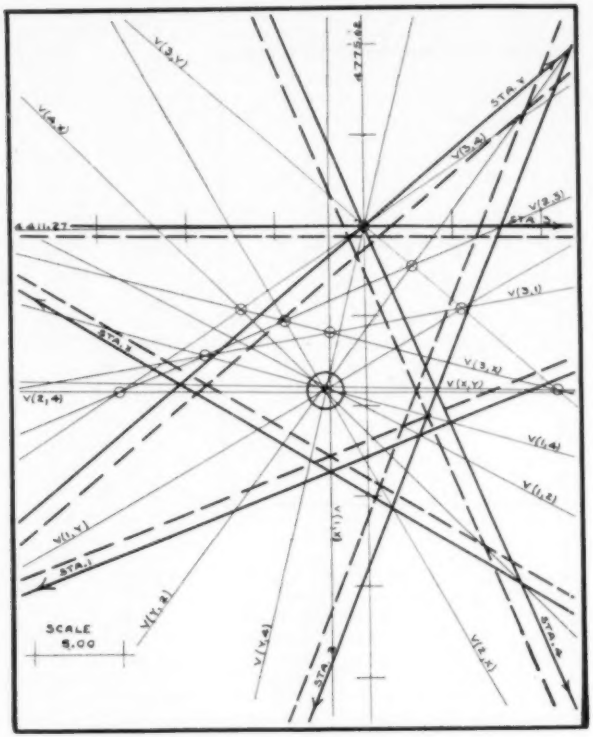

Fig. 10—Error figure for Station Z, using Stations 3, 4, and Y for the first solution.

Note that Station 3 is involved in all of the solutions falling outside of the heavily circled area.

Then (1) $E_B - E_A = (N_B - N_A) \tan \alpha$

(2) $N_B - N_A = (E_B - E_A) \cot \alpha$

The intersection of α with E_B is then

(3) $E_B = E_A + (N_B - N_A) \tan \alpha$

and the intersection of α with N_B is

(4) $N_B = N_A + (E_B - E_A) \cot \alpha$

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Developments in the

Application of Activated Carbon to Cyanidation

Including the Desorption of Gold and Silver from Carbons

by E. H. Crabtree, Jr.,
V. W. Winters, and
T. G. Chapman

This paper traces the experimental and pilot plant work completed by the authors since 1939 including the various methods which have been developed in applying coarse activated carbon to cyanidation. The methods and results developed by the authors in desorbing gold and silver from carbons are given and the possibilities for the re-use of the desorbed carbons are discussed.

IN 1939 one of the authors* described advances in carbon-cyanidation for the period 1932 to 1939 and included: (1) the dissolution of gold in an ore pulp by cyanide and its simultaneous adsorption by carbon, (2) the stage addition of adsorptive carbon and its movement countercurrent to the flow of the ore pulp, and (3) the recycling of adsorptive carbon. Considerable additional experimental and pilot plant work have been done by the authors since 1939 and the results of such work have been sufficiently encouraging to justify a second paper.

* T. G. Chapman: A Cyanide Process Based on the Simultaneous Dissolution and Adsorption of Gold. *Trans.* AIME (1939) 134, 207.

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Developments from 1939 to 1948: In 1941, Verne W. Winters did the first experimental work in eliminating agitation during dissolution and adsorption and thereby demonstrated that the dissolution of gold by cyanide and its simultaneous adsorption by carbon take place essentially to completion in a stagnant pulp. Shortly thereafter, and as a result of the stagnant pulp experimental work, Winters and Chapman demonstrated that soluble gold, in a cyanide ore pulp, would migrate from the ore pulp through a perforated container and be adsorbed by carbon within the container.

During the same year, the Eagle-Picher Mining and Smelting Co. erected a 25-ton capacity pilot plant at the Harquahala mine about 9 miles from Salome, Ariz. Experimental work at this plant comprised chiefly the stagnant pulp modification of carbon-cyanidation although considerable work was done in the grinding and activation of carbon, and testing various methods of extracting adsorbed gold from carbon.

Soon after the completion of the Harquahala

experimental work, the Eagle-Picher Mining and Smelting Co. erected a continuous, small capacity, pilot plant at its Sahuarita concentrator, about 20 miles from Tucson, Ariz., to continue experimental work with various modifications of carbon-cyanidation. This plant has operated almost continuously since this time.

At the conclusion of the war, the authors experimented with the Government war surplus, coarse gas-mask carbon. Several flowsheets employing such carbon were tried for the adsorption of dissolved gold from ore pulps including the stage addition and countercurrent movement of the carbon in pulps. Among these methods may be men-

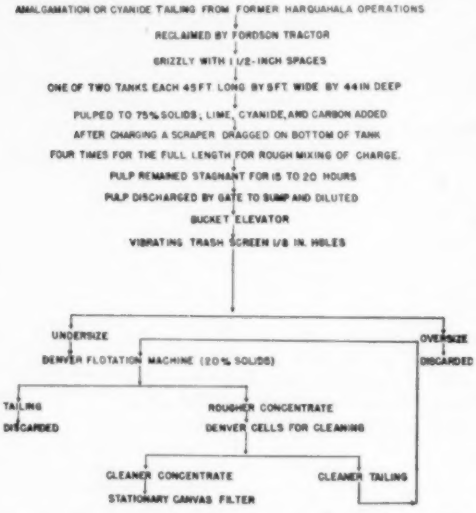
AMALGAMATION OR CYANIDE TAILING FROM FORMER HARQUAHALA OPERATIONS
RECLAIMED BY FORDSON TRACTOR
GRIZZLY WITH 1 1/2-INCH SPACES
ONE OF TWO TANKS EACH 45 FT LONG BY 5 FT WIDE BY 44 IN DEEP
PULPED TO 75% SOLIDS; LIME, CYANIDE, AND CARBON ADDED
AFTER CHARGING A SCRAPER DRAGGED ON BOTTOM OF TANK
FOUR TIMES FOR THE FULL LENGTH FOR ROUGH MIXING OF CHARGE.
PULP REMAINED STAGNANT FOR 18 TO 20 HOURS
PULP DISCHARGED BY GATE TO SUMP AND DILUTED
BUCKET ELEVATOR
VIBRATING TRASH SCREEN 1/8 IN. HOLES
UNDERSIZE
OVERSIZE
DISCARDED
DENVER FLOTATION MACHINE (20% SOLIDS)
TAILING
DISCARDED
ROUGHER CONCENTRATE
DENVER CELLS FOR CLEANING
CLEANER CONCENTRATE
CLEANER TAILING
STATIONARY CANVAS FILTER

Fig. 1—Stagnant pulp modification, Harquahala pilot plant.

tioned: (1) coarse carbon, ground to approximately 100 mesh was added to the ore pulp and separated from the pulp by flotation; (2) coarse carbon* was added to a stagnant ore pulp and separated from the pulp by screening; (3) coarse carbon was added to an agitated ore pulp and separated from the pulp by screening; and (4) perforated cylinders, containing coarse carbon, were placed in agitators and partially submerged into the ore pulps. The containers were slowly revolved for the adsorption of soluble gold and in a series of agitators the carbon, with or without the containers, was moved from agitator to agitator countercurrent to the flow of the pulp.

* The coarse carbon used for all methods described varied in size but usually was minus 10 plus 35 mesh.

During the progress of the work involving the use of coarse adsorptive carbon it was realized by the authors that if such adsorptive material were to be employed on a commercial basis either a lower cost carbon than that available during the testing period must be provided or the problem of desorbing the gold from the carbon and the re-use of the carbon must be solved. Referring to the availability of a lower-cost coarse carbon, although considerable progress has been made in reducing the cost, insofar as the authors are informed, the cost of such carbon is still too high to warrant employing it once and destroying it by burning for the recovery of its adsorbed gold. Recent work has demonstrated that carbon derived from ore pulps can be almost completely desorbed of its gold and silver and the carbon re-used for further adsorption.

It is believed, therefore, that a complete carbon-cyanide process, with several modifications, is now available which offers considerable promise from the standpoints of: (1) low capital cost of plant; (2) extractions, as compared to standard cyanide methods, are at least as high in the treatment of fast settling ores and higher in the treatment of clayey ores and some refractory ores; (3) the production of bullion without burning the carbon and smelting the resultant ash is feasible; (4) pregnant solutions resulting from desorption of carbon amount to only 10 to 15 tons per 100 tons of ore treated, thus greatly reducing the size of the gold precipitate unit; (5) a process which can treat slow-settling as readily as fast-settling ores; (6) the flotation modification recovers some forms of insoluble gold while floating the carbon and its soluble gold content; (7) during the last stages of adsorbing soluble gold it is not necessary to maintain free cyanide in the pulp whereas it is necessary to maintain free cyanide in solutions during conventional precipitation with zinc; (8) loss of efficiency in the adsorption stage due to foul solutions has not been encountered in carbon-cyanidation; (9) in the treatment of ores containing graphite or pyrrhotite, loss of soluble gold adsorbed on graphite and loss by premature precipitation by the decomposition products of pyrrhotite does not occur, at least not to the extent it does with standard cyanide methods; and (10) in the presence of certain cyanicides such as copper sulphide the stage addition of cyanide to maintain low concentrations of cyanide shows promise in reducing the consumption of cyanide.

The brief descriptions and data which follow have been taken from plant work completed since 1939 to indicate the progress which has been made with the various modifications of carbon-cyanidation.

Stagnant Pulp Modification: The Eagle-Picher Mining and Smelting Co., as previously mentioned, operated a 25-ton capacity pilot plant to confirm results which had been obtained in the laboratory in eliminating agitation during the dissolution of gold from an ore pulp by cyanide and its adsorption by carbon. The plant was erected at the Harquahala mine and operated on very slow settling clayey amalgamation or cyanide tailing from former operations. Fig. 1 gives a flowsheet of the Harquahala test plant. The reclaimed tailing was charged to one of two concrete tanks provided for the dissolving and adsorbing stages. Each tank was 45 ft long by 5 ft wide by 44 in. deep and held 25 tons of dry tailing as a pulp containing 75 pct of solids. The method of operating was as follows: The tailing was reclaimed and conveyed to a grizzly having 1½ in. spaces by a Fordson tractor equipped with Ferguson system hydraulic scraper control.

To fill a tank, 32 scraper loads, conveyed a distance of 150 to 200 yards, were required. Water, lime, and cyanide were first added to the tank after which one-third of the tailing was charged and activated carbon added to the surface. Tailing was again charged until the tank was two-thirds full and more carbon added to the surface. The final charge of tailing was then added with the remaining carbon added to the surface. The grizzly, shown in fig. 1, with scraper attached was then pulled back and forth twice for mixing the charge. The charge then stood, without further agitation, for a period of 15 to 20 hr after which the pulp was discharged to a sump and reclaimed by a bucket elevator. The bucket elevator discharged onto a 4x5 ft vibrating trash screen with ⅛ in. holes and the undersize was treated in Denver flotation cells for the separation of the carbon and its adsorbed values from the pulp.

Lime, cyanide, and carbon were added to the charge at the rates of 0.4, 0.5, and 3.6 lb per ton of tailing, respectively. Soft or hard wood charcoal, ground to from 48 to 150 mesh and activated at the plant by heating to 1400°F and cooling without quenching, was used.

Approximately 4000 tons of this material were treated on an experimental basis. The heads assayed from 0.085 to 0.11 oz gold per ton. The extraction varied from 66 to 79 pct and the carbon flotation concentrate assayed from 16 to 23 oz gold per ton. Barren solutions varied from 0.0042 to 0.0005 oz gold per ton depending upon the fineness of grinding the carbon and its adsorptive efficiency.

Table I gives typical results obtained when minus 150 mesh carbon was used and indicates results obtained with proper fineness of grinding but with rather crudely activated carbon.

Table I. Harquahala Test Plant Results, Tanks 129 to 137 Inclusive.

	Tons	Assay, Oz Gold per Ton	Gold, Oz	Gold, Pct
Heads	189.0	0.089	16.730	100.0
Carbon concentrate	0.5744	21.76	12.499	74.7
Barren solution	756.0	0.0009	0.680	4.1
Tailing	188.4	0.019	3.500	21.2

The following observations are offered with respect to the stagnant pool method of carbon-cyanidation and its potential usefulness.

1. The speed of dissolving and adsorbing gold without agitation is much faster than might be expected and is explained by the lower concentration of soluble gold surrounding the carbon particles as compared to the concentration of gold in other parts of the pulp. Similar to the flow of water in a pipe due to a difference in pressure, ions will migrate in an ore pulp if a difference in concentration is maintained between two parts of the pulp. The continuous adsorption of gold by carbon maintains this difference in concentration.

2. The results given in table I were obtained when a proper size but a rather inferior, cheap activated carbon was used. Experience obtained at the Harquahala test plant indicated that it was difficult to float carbon coarser than 100 mesh especially in the more dilute pulp of the cleaner circuit.

3. Due to the imperfect mixing of the charge, the assay of barren solutions varied to a considerable degree. It is believed that a very short agitation period, say 10 to 15 min, after the reclamation of the pulp from the stagnant pool and before flotation, would equalize irregularities due to imperfect mixing and yield not only more uniform barren solutions but also lower value barren solutions.

4. Steam activated carbon would, no doubt, have yielded lower barren solutions than the crudely activated carbon employed at Harquahala. It is questionable, however, that the higher cost steam-activated carbon would have been better from the economic viewpoint at the time the tests were made.

5. There is a potential field of use for such a method of carbon-cyanidation in the leaching of gold from tailing piles in place. If mechanical methods can be devised to pulp the surface of such tailing piles to a consistency of, say, 75 to 80 pct solids the reactions of dissolving and adsorbing gold will, it is believed, take place without agitation.

Flotation Modification: Flowsheets for the modification of carbon-cyanidation employing finely ground carbon for adsorption and flotation for the separation of the carbon from the ore pulp have been published.* This method may be used with one or more stages of adsorption. With multiple stage adsorption the carbon is moved countercurrent to the flow of the ore pulp.

* T. G. Chapman: A Cyanide Process Based on the Simultaneous Dissolution and Adsorption of Gold. *Trans.* AIME (1939) 134, 207.

There are at least two advantages to this method as compared to other modifications. First, a soft, rather crudely activated wood carbon may be used as the increased surface of the finer carbon apparently nearly compensates for the crude activation and, second, some minerals which contain gold, insoluble in cyanide, may be recovered in the flotation carbon concentrate.

Adsorption With Revolving Screens Containing Carbon: A general flowsheet of this modification is given in fig. 2. It should be noted, however, that the flowsheet, as given in fig. 2, may be changed to meet the requirements of the specific ore to be treated.

G. H. Roseveare, of the Arizona Bureau of Mines, working with Chapman in the laboratories of the University of Arizona was the first, to the author's knowledge, to construct a horizontal, revolving-screen test unit attached to a laboratory mechanical-type agitator. A few months later Chapman and others did the experimental work involving the permissible volume of carbon in a screen and the size of screen required in mill operation. A 5x5 ft Wallace agitator and a 12x15 in. revolving screen were used for this work.

This modification differs from other modifications in that the steps of dissolving the gold, adsorbing the dissolved gold, and separating the carbon from the ore pulp are combined and take place as a continuous process in a single step. It should be noted that the combining of the three steps was described for a stagnant pulp and that the revolving screen only served to make the batch process of the stagnant pulp modification a continuous process. The

countercurrent movement of the carbon may be used as in other modifications.

Referring to fig. 2, the data presented in this paper for this modification were obtained without the classifier in the grinding circuit and without the thickener following the grinding circuit. The reason for these changes in the flowsheet was the use of Harquahala tailing for testing which did not require grinding or thickening. The Sahuarita test plant flowsheet included a 16 x 32 in. Denver ball mill operated with a light ball load for pulping the tailing. The mill was fed at the rate of 60 lb of dry tailing per hour pulped to 37 pct solids. The ball mill discharged to a 20-mesh vibrating trash screen and the screen undersize was pumped to the first of three 36 x 36 in. Wallace agitators placed in series. Agitators 1 and 2 were each equipped with one 28-mesh stainless steel screen and agitator 3 had two screens. The screens, 9¾ in. in diameter and 5 in. long, were revolved at 35 rpm and were submerged approximately 2½ in. into the ore pulp. Each agitator provided a detention time of 8 hr. The countercurrent movement of the carbon was accomplished by moving the screens at 3-hr intervals from agitator 3 to 2 and from agitator 2 to 1. The carbon on leaving agitator 1 was desorbed and the desorbed carbon was recycled to the screens of agitator 3.

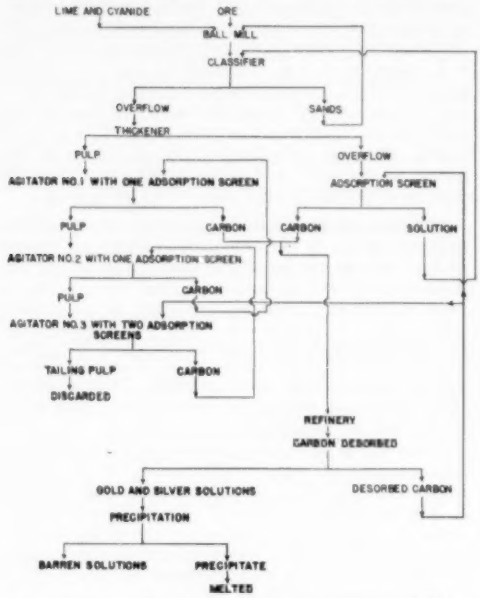

Fig. 2 — Adsorption by revolving screens containing carbon.

Lime, cyanide, and carbon were added at the rates of 0.73, 0.88, and 3.5 lb per ton of tailing treated respectively.

Table II gives the results of a 63-hr test with this flowsheet. The test was operated as described for 15 hr and for the remaining 48 hr a scavenging screen was added to the tailing launder. The tailing pulp was allowed to flow from the launder to the top of the revolving scavenging screen and a rounded depression in the launder permitted the screen some submergence in the pulp.

Table II. Container Modification, Results of 63 hr Test at Sahuarita Plant

	Weight, Lb.	Tons per 100 Tons of Heads	Assay, Oz Gold per Ton	Per Cent
Heads	3,780	100.0	0.096	100.0
Carbon	8.06	0.213	41.86[a]	92.7
Barren solution	6,426	170.0	0.0012[b]	2.1
Tailing	3,780	100.0	0.005	5.2

[a] Carbon assayed 29.7 oz silver per ton.
[b] Barren solution from agitator 3 averaged 0.0025 oz gold per ton.

Referring to table II the following comments are offered:

The barren solution given in the tabulation, namely, 0.0012 oz gold per ton, was the solution after treatment in the scavenging screen placed in the tailing launder. The barren solution from agitator 3 is given in footnote *b* as assaying 0.0025 oz gold per ton. The scavenging screen, with a very short time of contact with the tailing pulp, therefore reduced the assay of the barren solution from 0.0025 to 0.0012 oz gold per ton.

Desorption of Gold and Silver from Carbon: The desorption of gold and silver from a carbon that has been in contact with an ore pulp apparently depends upon the shifting of the equilibrium encountered between adsorption and desorption. Four methods of shifting the equilibrium succeeded in desorbing the gold and silver. The four methods comprised: (1) the use of a solvent in conjunction with a large excess of precipitant in order to remove the gold from the receiving solution as the desorption progressed; (2) the use of a solvent in conjunction with electrolysis to accomplish the same purpose; (3) the use of a solvent and large volumes of solutions, added in stages, to keep the concentrations of the gold and silver in the receiving solutions sufficiently low to accomplish desorption; and (4) the use of higher temperatures by employing hot solvent solutions in a pressure chamber. The authors believe that the last method given is the most feasible as this method is very rapid, has the lowest reagent cost, and accomplishes the desorp-

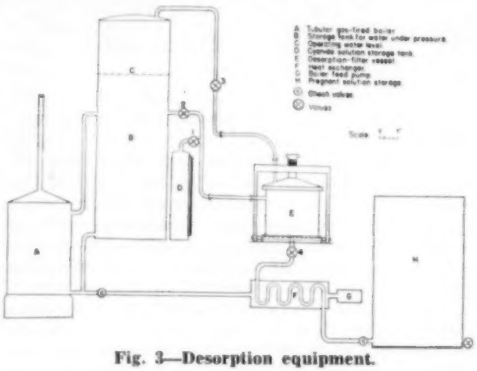

Fig. 3—Desorption equipment.

tion with the minimum volume of desorbing solution.

Fig. 3 gives the equipment, designed and constructed by Winters at the Sahuarita plant, for this method of desorption. The equipment consisted of a 1 hp, gas-fired, vertical fire-tube boiler *A*, and a hot water pressure storage tank *B*. The desorption was accomplished in the quick opening pressure filter *E*. Heat exchanger *F* and pregnant solution storage tank *H* completed the essential equipment.

The operating cycle comprised placing the carbon in the filter *E* and adding cyanide solution to the filter from tank *D* through valve *1*. Hot water was then added from *B* through valve *2* after which valve *2* was closed and valve *3* opened, which subjected the charge to the full boiler steam pressure of 90 to 95 psi. The cyanide used for each desorbing stage varied from 2 to 20 lb of 90 pct sodium cyanide per ton of carbon treated and the hot water for each desorbing stage amounted to 10 tons per ton of carbon. The soaking period was 20 min after which the closing of valve *3* and the opening of valve *4* forced the pregnant solution through the filter in *E*, through the heat exchanger *F* and into the pregnant solution storage tank *H*. The pipe which discharged the hot pregnant solution into *H* was kept submerged by solution constantly maintained in *H* to prevent flashing. Five cycles as described were used for each charge of carbon.

Table III. Desorption of the Carbon of Table II

	Carbon, Lb	Tons per 100 Tons of Mill Heads	Assay, Oz per Ton		Per Cent	
			Gold	Silver	Gold	Silver
Heads, carbon, screens 11 to 31	8.06	0.213	41.86	29.7	100.0	100.0
Pregnant solution	466.	12.32	0.68	0.48	98.4	100.0
Desorbed carbon	8.06	0.213	0.63	Trace	1.6	0

Table III gives the data for the desorption of the carbon used in the 63 hr test of Table II.

Referring to Table III, the desorption of the gold was 98.4 pct and the desorption of the silver was essentially complete. The volume of the pregnant gold and silver solution, including the diluting water added to the pregnant gold solution storage tank to prevent "flashing," amounted to 12.32 tons per 100 tons of ore treated which is equivalent to 57.8 tons of pregnant solution per ton of carbon desorbed. The pregnant solution assayed 0.68 oz of gold and 0.48 oz of silver per ton. If the pregnant solution storage tank was operated on a continuous basis, as would be the case in practice, dilution to avoid flashing would not be necessary and the volume of the pregnant solution would be 10.65 tons per 100 tons of ore treated and 50 tons per ton of carbon desorbed.

Assuming an average settling ore with average size of grinding treated by countercurrent decantation, the volume of pregnant solution for precipitation would amount to approximately 300 tons per 100 tons of mill heads. The solution precipitated by standard cyanide methods would therefore be approximately 28 tons as compared to one ton of solution produced by the desorption of carbon.

The desorption of the activated petroleum coke used in this test was not as complete as that obtained with shell carbon as given in table IV. In table III the percentage of gold desorbed was 98.4 for the petroleum coke as compared to an average desorption of 99.5 pct for shell carbon.

Cyanide was added for the test described at the rate of 2 lb per ton of carbon per cycle or a total of 10 lb of cyanide per ton of carbon. With other carbons it was necessary to add greater amounts of cyanide. Tests to determine minimum cyanide requirement for a given carbon have not been made to date.

The maximum capacity of the 10 x 12 in. desorption chamber shown in fig. 3 has not been determined. As operated in tests described in this paper one charge of carbon amounted to 2.22 lb and five 20-min soaking periods were used per charge. Assuming a total time of 2 hr for desorption the capacity would be 26.6 lb of carbon per 24 hr. This amount of carbon would be obtained from the treatment of approximately 7.5 to 9 tons of low grade ore. The soaking time per cycle can undoubtedly be decreased so that the capacity given is probably low.

Comparison of Desorbing Petroleum Coke and Shell Carbon: Table IV gives the comparative results of desorbing activated petroleum coke and

Table IV. Desorptions of Petroleum Coke and Shell Carbon Compared

Nos. of Screens* for Adsorption	Kind of Carbon	No. of Previous Desorptions	Wt. of Carbon Desorbed, Gram	Assays of Carbon Heads to Desorption, Oz per Ton		Assays of Desorbed Carbon, Oz per Ton		Per Cent Desorbed	
				Gold	Silver	Gold	Silver	Gold	Silver
16 to 22 incl.	Petroleum coke	None	1,008	42.76	33.6	0.33	None	99.2	100.0
23 to 29 incl.	Petroleum coke	57 pct fresh, 43 pct desorbed once	1,008	39.76	19.2	0.46	None	98.8	100.0
11 to 15, 30, 31	Petroleum coke	70 pct fresh, 30 pct desorbed once	1,008	43.07	37.2	1.12	None	97.4	100.0
40 to 44 incl.	Shell-Sahuarita activated	None	720	55.44	53.8	0.24	None	99.6	100.0
45 to 49 incl.	Shell-Sahuarita activated	None	720	41.01	49.4	0.138	0.55	99.6	98.9
50 to 56 incl.	Shell-factory activated	None	1,008	40.71	70.6	0.238	2.6	99.4	96.3
65 to 67 incl.	Shell-factory activated	1	432	46.33	52.6	0.27	0.68	99.4	98.7

* The screen numbers refer to the screens used for adsorption in the container modification (see fig. 2 and table II), and are given only to identify the carbons used for desorption.

Table V. Recycling of Desorbed Carbons

Nos. of Screens*	Time of Test, Hr	Kind of Carbon	New, Desorbed, or Reactivated Carbon Used	Assay of Carbon, After Adsorption, Oz per Ton		Barren Solution Assays, Oz per Ton			
						After Third Agitator		After Scavenging Screen	
				Gold	Silver	Gold	Silver	Gold	Silver
11 to 26	48	Petroleum coke	New, factory activated	43.72	30.8	0.0022	0.007	0.0010	0.004
27 to 33	21	Petroleum coke	Screen 11 to 26 desorbed	37.15	29.4	0.0036	0.010	0.0018	0.006
34 and 35	6	Petroleum coke	Screens 27 to 33 desorbed	36.44	40.4	0.0042	0.010	0.0020	0.005
36 to 39	12	Petroleum coke	Desorbed twice, Sahuarita reactivated	44.35	65.5	0.0032	0.012	0.0011	0.006
40 to 49	30	Shell carbon	New, Sahuarita activated	48.22	51.6	0.0025	0.011	0.0009	0.006
57 to 64	24	Shell carbon	Screens 40 to 49 desorbed	31.37	38.2	0.0028	0.012	0.0014	0.008
50 to 56	21	Shell carbon	New, factory activated	40.71	70.6	0.0017	0.007	0.0008	0.005
65 to 67	9	Shell carbon	Screens 50 to 56 desorbed	46.33	52.6	0.0035	0.013	0.0020	0.010

* The screen numbers refer to the screens used for adsorption in the container modification (see fig. 2 and table II), and are given only to identify the carbons used.

shell carbon. The average desorption of gold and silver from petroleum coke was 98.5 and 100 pct, respectively, as compared to 99.5 and 98.5 pct for shell carbon.

Although the results given in Table IV show some increase in the assays of the desorbed residues for the second desorption of the same lot of carbon, the increase is not serious and the repeated desorptions of other carbons, not reported in this paper, did not indicate any build up of gold and silver in the residues after repeated desorptions.

Table V gives a comparison of barren solutions and assays of carbon obtained when freshly activated, desorbed, and desorbed reactivated carbons were employed for adsorption in the container modification.

Referring to the barren solutions, two barren solutions are included in the tabulation. The first is the solution discharged from the third agitator, and the second, the solution leaving the scavenger screen of the tailing launder. The figures in the tabulation show an increase in gold content of the barren solution with petroleum coke and shell carbon in the recycling of the desorbed carbon without reactivation. With petroleum coke, the fresh activated carbon yielded a barren solution of 0.0010 oz gold per ton after the scavenging screen, a barren solution of 0.0018 oz with the recycling of the carbon after one desorption, and a barren solution of 0.0020 oz with the recycling of the carbon after two desorptions. The ratio of solution to solids leaving the scavenger screen was 1.7 to 1. With the reactivation of the carbon, however, after two desorptions the barren solution was reduced to 0.0011 oz. The data in the tabulation is not as complete for the shell carbon but does show an increase in the assay of the barren solution with recycling after one desorption.

The assay values of barren solutions depend not only upon the adsorptive efficiency of the carbon but also upon the kind and length of contact of the carbon with the ore pulp. For example, comparing the barren solutions of the third agitator with those of the scavenging screen it is noted that improved contact between the carbon within the screen and the ore pulp drastically reduced the barren solution loss.

Although the data show that desorbed carbon, which has not been reactivated, has decreased adsorptive speeds it is entirely possible that more effective contact with such carbon will make up to a considerable degree for the loss of adsorptive speed for such carbons. The data indicate that the reactivation of desorbed carbon restores its adsorptive speed to that of fresh carbon.

The authors believe that the loss of adsorptive speed of a desorbed carbon as compared to a fresh carbon is due to the partial filling of the interstitial spaces of the carbon by slime. The reactivation of desorbed carbon, which apparently restores its adsorptive speed, is believed to increase the interstitial spaces by oxidation with steam during reactivation.

Summary: This paper describes the use of the carbon-cyanidation process in a stagnant ore pulp in a pilot plant having a capacity of 25 tons per day. Minus 150-mesh activated carbon, with lime and cyanide, were pulped with gold-bearing tailings residues at a consistency of 75 pct solids and allowed to stand for a period of 15 to 20 hr. The carbon was then recovered from the diluted pulp by flotation. About 75 pct of the gold was recovered in the carbon flotation concentrate.

Another method using coarse activated carbon in revolving screen containers partially immersed in a series of three agitators recovered 92.7 pct of the gold in a coarse carbon concentrate assaying 41.86 oz of gold per ton of carbon. This concentrate was then treated by a desorption method described in which 98.4 pct of the gold was extracted in a pregnant solution assaying 0.68 oz of gold per ton of solution. The volume of this solution amounted to only 12.32 tons per 100 tons of ore treated. The desorbed carbon was re-used in succeeding tests.

Comparative desorption tests on activated petroleum coke carbons and shell activated carbons indicated that gold desorption of the shell was slightly better, reaching as high as 99.6 pct of the gold. Tests indicated that the adsorptive efficiencies of the carbons after desorption were not quite as good as originally, but that after two desorptions followed by reactivation, the adsorptive speed was practically restored to that of fresh carbon.

Further work is required to determine just how many times the carbon may be re-used.

Continuous Countercurrent Decantation Calculations

by T. B. Counselman

Continuous countercurrent decantation calculations have always been a headache to the cyanide man (and the chemical engineer) because of the simultaneous equations involved. These are tedious to solve, and there is considerable opportunity for mathematical error. This paper presents a shortcut method for making these calculations, which has been in successful use for twenty years by the author, and which is particularly valuable in selecting the best of several possible flowsheets, or evaluating the effect of various assumptions.

EVERYONE who has to calculate cyanide circuits, using either thickeners, filters or both, realizes the headaches involved in solving a set of simultaneous equations. When you calculate a large number of these, based on experimental data, trying to arrive at the best flowsheet, you can spend days at the task.

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AIME New York Meeting, Feb. 1950.

TP 2783 B. Discussion (2 copies) may be sent to Transactions *AIME before March 31, 1950. Manuscript received Aug. 23, 1949.*

Herewith is presented a short method, equally accurate with the standard simultaneous equation method, and much quicker.

Here for instance is the standard simultaneous equation method, for 100 tons per day of $10 ore, with the conditions assumed as below (fig. 1).

Conditions Assumed

a. 100 tons of ore per day crushed in cyanide solution.

b. Discharge from all thickeners with 50 pct moisture.

c. $10 value dissolved per ton of ore.

d. 50 pct in mill and 50 pct in agitator.

e. 400 tons of solution from thickener *V* precipitated to $0.02.

f. Agitation with a dilution of 2 of solution to 1 of solids.

g. Let *V*, *W*, *X*, *Y* and *Z* represent value in dollars per ton of solution discharged from the respective thickeners.

Equating Out of and Into Each Thickener

1. $100V$ plus $400V = 500W$ plus $(0.50 \times \$10 \times 100)$
2. $100W$ plus $600W = 500X$ plus $100W$ plus $(0.50 \times \$10 \times 100)$ plus $100V$
3. $100X$ plus $500X = 100W$ plus $500Y$
4. $100Y$ plus $500Y = 100Z$ plus $100X$ plus (400×0.02)
5. $100Z$ plus $100Z = 100Y$ plus 100 tons of water (value $0.00)

Simplifying:

1. $V = W$ plus 1.00
2. $W = X$ plus 1.20
3. $X = Y$ plus 0.24
4. $Y = 0.2Z$ plus 0.064
5. $2Z = Y$

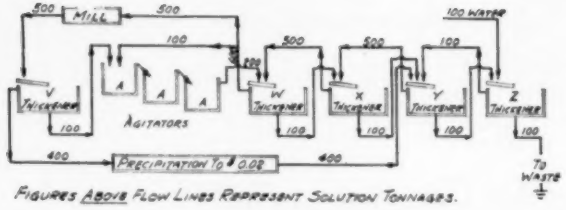


Fig. 1—Simultaneous equation method.

Solving:

$V = \$2.51111$
$W = \$1.51111$
$X = \$0.31111$
$Y = \$0.07111$
$Z = \$0.03556$ —*Loss* per ton

To Check the Calculations Above:

The amount precipitated from 400 tons at \$2.5111—0.02	= \$996.44
The amount lost in tailings, 100 tons at \$0.03556	= 3.556
	\$999.996

The amount dissolved, 100 tons at \$10.00 = \$1,000.

From the Foregoing, the Following Results are Deduced:

Assay value of pregnant solution, i.e., Value of *V*	= \$2.51111
Assay value of discharged solution, i.e., Value of *Z*	= 0.03556
Loss of dissolved value per ton of ore	= 0.03556

Dissolved value saved, 99.64 pct

Shortcut Method

Now for the shortcut method, using exactly the same assumptions. First sketch the flowsheet exactly as before (fig. 1), and put the solution tonnages above the line exactly as before (fig. 2).

The conditions assumed are the same as before.

Now assume that the total dissolved value loss per day is (x) dollars. In other words, the dissolved value in the tailing discharged as underflow from thickener *Z*, is (x). Put (x) under the line of the tailing from this thickener. Since the concentration of dissolved value must be exactly the same in the overflow and underflow of any thickener, and since the solution tonnage of overflow and underflow is the same in this case, it follows that the dissolved value in the overflow of thickener *Z* is also (x). Therefore, put (x) under the line of the overflow of this thickener.

The total discharge from thickener *Z*, overflow plus underflow, contains dissolved value in the amount of $(2x)$. The wash water added to *Z* is barren. Therefore, this total of dissolved values of $(2x)$ must come with the underflow of thickener *Y*. Therefore, put $(2x)$ under the line of the underflow from *Y*.

Since the concentration in the overflow must be the same as in the solution with the underflow, but since the tonnage of overflow is in this case, 500, as against 100 tons of solution in the underflow, it follows that the dissolved value in the overflow of thickener *Y* is $(10x)$, and this figure is put under the line of the overflow.

Leaving thickener *Y* therefore, we have $(10x)$ with the overflow and $(2x)$ with the underflow or a total of $(12x)$. Coming into this same thickener is the overflow of thickener *Z*, plus the solution after precipitation which carries a total value of \$8, (400 tons of solution, precipitated to two cents). The only other source of values coming to thickener *Y* is the underflow of thickener *X*. The value of its underflow is $(10x)$ plus $(2x)$ — (x) — \$8 = $11x$ — 8, which we write under the underflow line of thickener *X*.

By the same method of proportioning, we find that the overflow of thickener *X* contains values amounting to $(55x - 40)$ and that the underflow of *W* must contain $(56x - 48)$. By the same method we find that the solution going to the mill carries $(280x - 240)$, and the 100 tons of overflow from *W* which goes to the agitators carries values of $(56x - 48)$. Also the discharge from the agitators to thickener *W* must carry $(337x - 296)$.

Knowing what comes out of the agitators, $(337x - 296)$, and the portion of *W* overflow carrying $(56x - 48)$ added to the agitators, the value of the solution with the underflow of thickener *V*, *after all gold values are dissolved*, must be $(281x - 248)$. However, we have assumed 50 pct of the gold dissolved in the mill, and 50 pct, or \$500, dissolved in the agitators. Therefore, we must subtract another 500 from the value in the solution with the underflow from *V* because as yet the gold is undissolved. This makes the solution value of the underflow of *V* $(281x - 748)$.

By proportion, the value in the overflow, going to precipitation is $(1124x - 2992)$.

Now then, the total dissolved values in the overflow plus underflow of thickener *V*, with half the gold as yet undissolved, less the dissolved value of the solution going to the ball mill must equal the \$500 assumed to be dissolved in the mill. Or:

$$(1124x - 2992) \text{ plus } (281x - 748) - (280x - 240) = \$500$$
$$1125x - 3500 = 500$$
$$(x) = 3.556 \text{ dissolved loss per day}$$

The dissolved loss per ton is of course \$0.03556, which checks exactly with the simultaneous equation answer, but only one unknown and only one equation had to be solved.

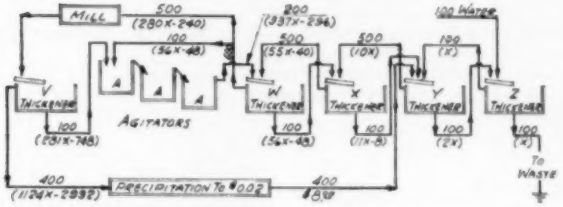
Checking this:

Total value dissolved in plant, 100 tons at $10 (50 pct in mill) (50 pct in agitators) =	$1000.00
400 tons precipitated: (1124x — 2992) — 400 times $0.02 = 1004.94 — 8.00	996.944
Loss or residue (x)	3.556
Total	$1,000.500

back to thickener Z, (displacing an equal amount of wash water previously added to that thickener), and the stronger filtrate back to thickener Y, the tonnages would divide as follows (fig. 3):

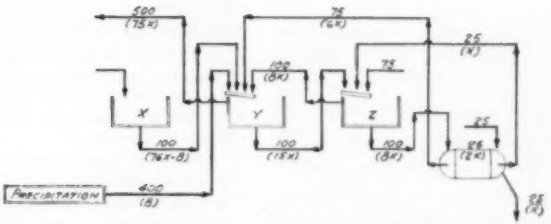
The (x) values balance out, by the method described above, as shown for the last part of the flowsheet.

It can, of course, be assumed that there will be a volumetric displacement of, say, 75 pct of the solution in the filter cake, followed by dilution of



FIGURES ABOVE FLOW LINES REPRESENT SOLUTION TONNAGES
FIGURES ( ) BELOW FLOW LINES ARE TOTAL VALUES PER 24 HRS.

Fig. 2—Shortcut method.



FIGURES ABOVE FLOW LINES REPRESENT SOLUTION TONNAGES

Fig. 3—Shortcut method with filter.

The above flowsheet can be made and the loss calculated in a few minutes, as against 20 or 30 min for simultaneous equations, and gives satisfactorily accurate results.

The same simplified method can be used to include filtration, at any desired point in the flowsheet (fig. 3). For instance in the example given above, if the final thickener underflow were filtered to a final cake moisture of, say, 25 pct, you would assume that the filtration took place in two steps. First the straight filtration to final moisture, and second the addition of an amount of wash water equal to the moisture left in the cake, with a second filtration to final cake moisture. Conventionalized for calculations, and taking the wash filtrate the remaining solution with water. The same principle of computation would apply to whatever assumption was selected.

This same method of calculation may be used for calculating countercurrent decantation circuits in the chemical industry, for example causticizing flowsheets, the manufacture of alum from bauxite, or the washing, by countercurrent methods, of any solid, free from solutions.

Acknowledgment

Thanks are due to Mr. Dycus, of Golden Cycle Corp., and Nathaniel Herz, of Homestake Mining Co., for helping to work out the application of this method of calculation to cyaniding.

Effect of Conditioning on Flotation of Chalcocite

Chalcocite flotation is affected by agitation intensity during conditioning. Sodium sulphide in minute quantity may activate or depress, depending on conditions. Oxygen is a depressant while oxygen and sulphide neutralize each other by reaction at the mineral surface. Diffusion across a stagnant film at the surface governs reaction rates, while film thickness is reduced upon increase of agitation intensity.

by S. B. Tuwiner and S. Korman

Introduction

THE purpose of this study is to investigate the effect of agitation upon the flotation of chalcocite. It is believed that the concentration of collector which is required to produce a water-repellent surface upon the mineral is determined among other things by the rate of collector diffusion through an impeding film of more or less stagnant fluid. Its thickness depends upon the intensity of agitation.

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AIME Columbus Meeting, September 1949.

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In the first part of this study a method is developed for determining the thickness of the fluid film upon a polished surface of synthetic chalcocite. The film thickness is then determined as precisely as possible under three readily reproducible conditions of agitation intensity. These determinations serve as a quantitative evaluation of agitation.

In Part 2 it is demonstrated that in solutions of lime at pH 10.5 and in the absence of air or dissolved oxygen the concentration of collector required to develop water-repellency upon the surface in any given time is a function of the fluid film thickness, and that after about 10 min the relation becomes linear. This points to the possibility that the collector ions may be involved in a diffusional process through the fluid film.

In Part 3 it is shown that where air is present more collector is required to produce water-repellency than for the case in which air is excluded. Next, it is shown that for the latter case the most minute amount of sodium sulphide suffices to preclude development of water-repellency. Both oxygen and sulphide are in a sense, therefore, depressants for chalcocite in a lime solution at pH 10.5. When oxygen and sulphide are present simultaneously there is a region of minimum critical collector concentration for water-repellency which is very markedly lower than the concentration where oxygen alone or where sulphide alone is present. Moreover, this minimum appears to be identical with the collector concentration which is required where sulphide and oxygen are both excluded. This would appear to indicate that in certain ranges of concentration of oxygen and sulphide, each cancels the effect of the other. The range of concentration of solution for this effect in solutions saturated with oxygen is a function of the agitation intensity. Thus we are led to believe that a chemical reaction involving sulphide and oxygen occurs at the surface of the chalcocite.

Part 4 of this study is a further development of this idea and some of its consequences from an electrochemical standpoint, while in Part 5 the disappearance of sulphide from pulps of several minerals is noted with an indication that both dissolved oxygen and the mineral are necessary, the reaction being catalytic or pseudocatalytic.

Part 1. Determination of Fluid Film Thickness

The Fluid Film Concept: The hindrance to the diffusion of collector to the surface of the chalcocite is a barrier of stagnant fluid in the vicinity of the surface. It is the same barrier that limits the rate of transfer of heat, the rate of solution of dissolving crystals, or any of the other rates pertaining to chemical engineering unit operations in which the film resistance to diffusion plays a principal role.[1] Fluid films are present upon the surface to an extent which varies with the geometry and intensity of agitation. They result from the fact that in the region within the fluid near a surface, motion is entirely parallel to it. From the standpoint of transfer to or away from the interface the effect is as though the fluid were stagnant. Outside of the film, movement of the fluid may be streamline or turbu-

lent. Transfer is here principally by convection and usually very much more rapid than across the fluid film, in which diffusional processes are controlling. If a soluble substance such as a flotation collector is present in low concentration in a flotation pulp, and if this substance is consumed continually at the interface, then it is conceivable that its concentration near this interface may approach zero. The solute will continuously diffuse through the film

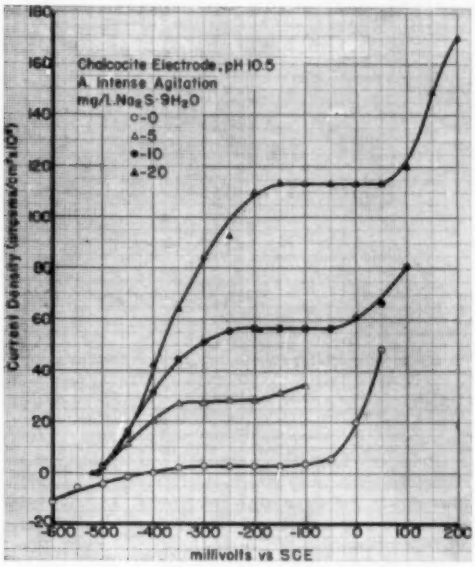


Fig. 1—Sulphide diffusion in air-free solution.

from the bulk of the solution, in which rapid transfer by convection predominates. The rate of its transfer through the film will be proportional to the concentration difference, and inversely, to the length of path traversed, the effective thickness of the fluid film.

Under conditions of quiescence this film will be comparatively thick and the diffusion relatively slow. On the other hand, in an intensely agitated pulp the stagnant film must be much thinner, and the diffusional rate consequently greater. In numerous chemical engineering studies these very elementary and basic considerations have resulted in an effective correlation of such physical factors as velocity, viscosity, and density with the rates of fluid flow, heat transfer, and material transfer in a rational and consistent manner. The same kind of analysis can be made as fruitful for the design of processes and equipment in the art of flotation as it has been for boilers, evaporators, condensers, distillation columns, and other equipment.

Method of Study: From this investigation it is shown that the collector concentration which is needed to give a water-repellent surface in any specified time is very greatly affected by the intensity of agitation. From this it is inferred that surface phenomena involving collector as well as other soluble substances which affect the flotation process are hindered by a diffusional resistance. It is assumed that there exists upon the solid surface a layer of fluid through which material must pass by a relatively slow process of diffusion, moving always from a region of higher to one of lower concentration either to or away from the interface. This film is never observed directly but is manifest only through the properties which it possesses, its effective thickness being determined in any case by a study of some rate process of material transfer.

In this study the effect of agitation intensity upon the conditioning of chalcocite is observed in an alkaline solution containing known amounts of potassium n-butyl xanthate and of Reagent 208. A specimen of synthetic chalcocite is mounted in lucite and polished in the usual way. Three degrees of agitation intensity are selected for the conditioning of this specimen. For the first degree, designated as "intense agitation," the specimen is held face down against the stream produced in a Waring Blendor. For the second, termed "moderate agitation," the chalcocite surface is left face up in a glass

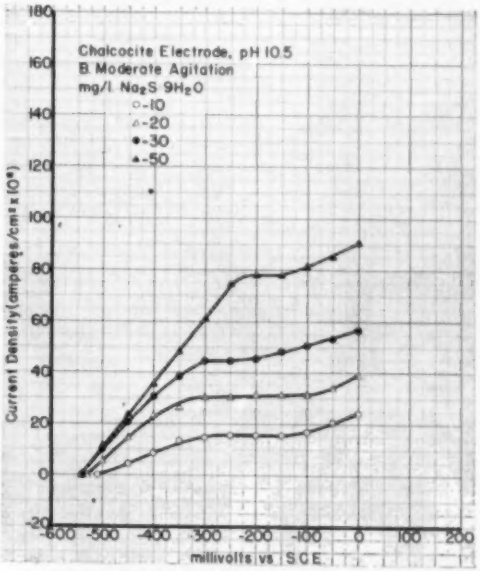


Fig. 2—Sulphide diffusion in air-free solution.

cell such as is used for contact angle measurement, agitation being produced by means of a stream of gas or air passing upward through the solution from a porous stone diffuser. The third degree of agitation is obtained in the same cell but under the "stagnant" condition which has become customary for contact angle study.

The terms "intense," "moderate," and "stagnant" have only qualitative significance; a quantitative measure of intensity is preferable. For this the effective fluid film thickness obtained by measuring the rate at which sulphide ions from solutions of known concentration are oxidized at a chalcocite

electrode by means of positive electric current. In these experiments, air is excluded. Under each condition of agitation and for several concentrations of sodium sulphide, increments of positive current were passed through the mineral, the cathode being a platinum wire electrode. The electric potential was noted for the chalcocite anode referred to a saturated calomel electrode.

Experimental Details: *Measurement of Sulphide Diffusion Currents:* For measurement of the diffusion current during intense agitation in the Waring Blendor the polished mineral specimen is mounted face down and held in position by means of an insulated ⅛ in. diam brass rod which is screwed into the back of the lucite mounting to make electrical contact with the chalcocite. The assembly is sealed into the cover of the Blendor. The cover is also provided with a sealed entry tube for pure hydrogen gas and a sealed platinum cathode wire. The saturated calomel electrode is clamped in fixed position through a loose fitting hole in the cover, which also provides for exit of gas.

The specimen, after preliminary polishing with metallographic paper, is polished with levigated

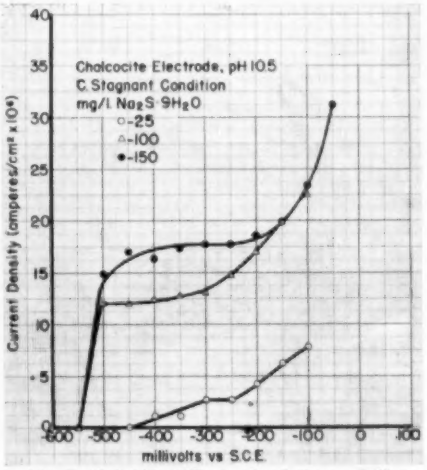


Fig. 3—Sulphide diffusion in air-free solution.

alumina and rinsed thoroughly with demineralized water. It is then placed in lime water, pH 10.5, containing 0.004 mol sodium sulphide, where it is held for a period of 1 hr. The Blendor is filled with 800 cc of lime water at pH 10.5 containing a specified concentration of sodium sulphide. Commercial tank hydrogen passed through copper turnings at 600°C is flushed through the solution for 10 min. The chalcocite-cover assembly is installed upon the Blendor and secured in position with rubber tape. Hydrogen gas is allowed to flow continuously at a rate of 1 liter per min during the course of the current potential measurements as described below.

For moderate agitation or stagnant condition, the chalcocite specimen is polished and conditioned in sodium sulphide solution as described above. Electrical contact is obtained by means of an insulated threaded shaft at right angles through the lucite to the side of the specimen. The chalcocite specimen is then placed face up in a solution of 300 cc of lime water at pH 10.5 containing specified concentration of sodium sulphide at the bottom of a 400 cc capacity, rectangular, glass cell similar to that commonly used in contact angle measurements. The same platinum cathode and saturated calomel electrode are installed and held rigidly by clamps. A cylindrical stone diffuser 1 x ⅞ in. diam for hydrogen is

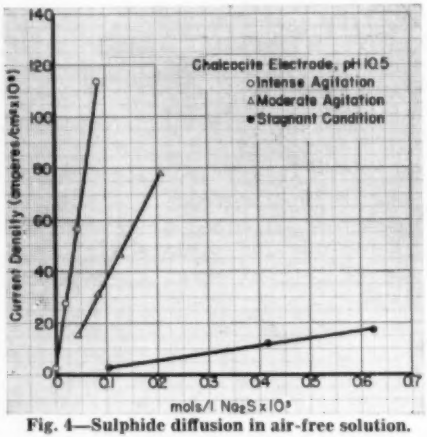


Fig. 4—Sulphide diffusion in air-free solution.

introduced to the bottom of the cell. A flow rate of 1 liter per min, as measured with a laboratory flowmeter, produces a reproducible degree of agitation and at the same time removes all oxygen from the solution and prevents its further access. For the stagnant condition, following preliminary flushing of the solution with hydrogen for 15 min, the diffuser is raised until it is completely out of the solution but is maintained at the top of the cell above the solution during the course of the measurements. Great care must be taken to insure that the diffuser for this type of conditioning does not disturb the surface of the solution, since it was found that the slightest turbulence at the surface is sufficient to provide a very appreciable increment in the diffusion currents obtained.

Current Potential Measurements: Current is supplied from a dry cell battery source across which is a 180 ohm slide wire potentiometer. That portion of the current which is supplied to the chalcocite surface is controlled by the position of the tap. The circuit is completed through a Weston microammeter with the positive lead to the chalcocite and the negative lead to the platinum cathode. Potential measurements are obtained by means of a Beckman industrial type potentiometer in circuit with the chalcocite and the saturated calomel electrode.

Current values are obtained by adjustment of the position of the slide wire tap and read on the microammeter. Corresponding potentials are read directly on the Beckman potentiometer.

Sulphide Diffusion and Film Thickness: A number of curves expressing the current-potential relations are shown in figs. 1, 2, and 3. Significantly,

there is a potential in all cases corresponding with the plateau region of the curves in which primary reaction involving the oxidation of sulphide ions appears to have reached its upper limit, the amount of electric current in each case being determined by the rate at which the sulphide ions are able to diffuse through the fluid film resistance. As is shown in fig. 4, these so-called diffusion currents are precisely proportional to the sodium sulphide concentration, indicating that the driving force of this

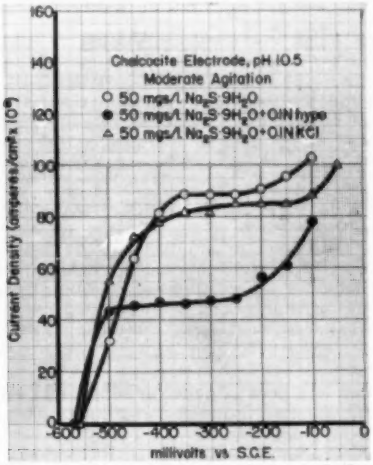


Fig. 5—Sulphide diffusion in air-free solution.

material transfer process is the concentration gradient across the fluid film which has its highest value at the potential corresponding with the plateau regions of the current-potential curves at which presumably the concentration of sulphide at the mineral interface has become zero. To compute the effective fluid film thickness in each of the three conditions of agitation the following formula[7] is used:

$$l = \frac{D}{i/A}\, n F C$$

Where l is the mean fluid film thickness in cm, i/A is the current density in amperes per cm². D is the diffusion coefficient for sulphide ions in cm² per sec, and n is the number of electrons transferred in the oxidation reaction from each sulphide ion. F is the faraday, equal to 96,500 amp-sec, and C is the concentration of sulphide in the solution in gram molecules per cm³. All of these quantities are known from our measurements, except the diffusion coefficient for sulphide or hydrosulphide, D. This is not given in the literature, although 1.4×10^{-5} cm² per sec has been given as the approximate diffusion coefficient for H_2S at 15°C.[8] Assuming this value for the diffusion coefficient the following values of the mean fluid film thicknesses are computed: 41, 147, 1940 microns for the three conditions of agitation studied. In calculating these it is assumed that n in the equation above is equal to 4, corresponding to the chemical reaction at the electrode surface of

$$2\,HS^- + 3H_2O \rightarrow S_2O_3^{=} + 8H^+ + 8e^-$$

The following facts support this assumption:

As shown in fig. 5 the addition of potassium chloride equivalent to 0.1 mol does not appreciably alter the diffusion current in these measurements. The effect of KCl is to reduce the voltage (IR) drop between electrodes. This would indicate that the currents actually are composed entirely of diffusion currents and that the so-called migration current is not included. Adding to these solutions sodium thiosulphate in the amount of 0.1 normal has the effect of reducing these diffusion currents by approximately one-half as shown in fig. 5. This effect of thiosulphate is the product of the sulphide oxidation and that by an excess of thiosulphate the reaction is altered from that shown above to the following:

$$2HS^- \rightarrow S^{\circ} + 2H^+ + 4e^-$$

The following considerations are taken into account in determining the most likely electrode process for the oxidation of sulphide: In the first place, the fact that linear relations are obtained between maximum diffusion current density and concentration of sulphide, shown in fig. 4, is a strong indication that the electrode reaction leads to a single sulphide oxidation product. Furthermore, it is unlikely that the diffusion currents measure the formation of oxidation products such as sulphite, pentathionate, and tetrathionate, all of which react directly with sulphide ion. Again, in view of the fact that thiosulphate ion does not produce a diffusion current in the range of potential studied, indicates that more extensive oxidation to di- and trithionate ion does not occur. Thus, oxidation to thiosulphate represents the greatest extent of oxidation possible as a single product which is not present in excess. When thiosulphate is present in excess, the diffusion current is reduced to half value, indicating a shift to a process forming a still lower oxidation product, namely, sulphur.

A flow of positive current to the mineral surface is used to oxidize dissolved sodium sulphide. This current (assuming no side reactions) would be proportional to the rate of diffusion, and discharge in accordance with Faraday's law. Rate of diffusion would in turn depend upon the potential imposed upon the mineral for it is this potential which determines the concentration of sulphide in the solution at the interface. At a potential at which this concentration becomes zero any further change can have but slight effect inasmuch as it does not further augment the concentration gradient of sulphide across the stagnant film. It is the concentration drop, the driving force, which is responsible for the diffusion, and thereby for the current.

Meaning of Diffusion Currents: The phenomenon illustrated in figs. 1, 2, and 3 is a result of an extreme state of concentration polarization which results from the depletion of sulphide at the electrode surface by the electrode reaction. The limit to the diffusion current which is marked by the plateau region of the graph signifies that sulphide ions are oxidized as rapidly as they reach the chalcocite electrode surface and their concentration at this surface remains constant at a value which is negligi-

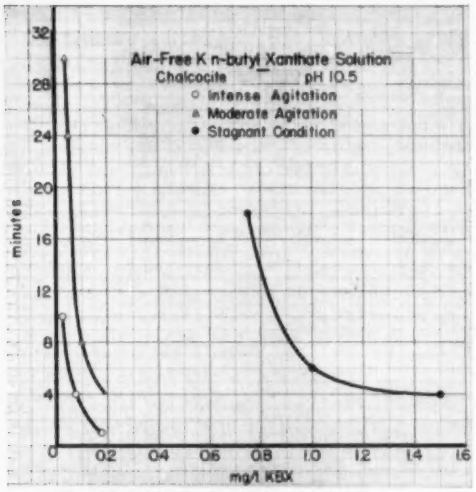


Fig. 6—Critical collector concentration vs. conditioning time.

bly small compared to the concentration in the body of the solution.

Under these conditions the current resulting from the electrode reaction is independent within certain limits of the applied e.m.f. and is governed solely by the rate of supply of the oxidizable substance to the electrode surface from the surrounding solution. In the most general case, reducible or oxidizable ions are supplied to the depleted region at the electrode surface by two forces: (1) a diffusive force proportional to the concentration gradient at the electrode surface, and (2) an electrical force proportional to the electrical potential difference between the surface and the solution. Reducible or oxidizable ions are therefore supplied to the electrode surface partly by diffusion and partly by electrical migration so that the limiting current can be regarded as the sum of a diffusion current and a migration current.[4]

It is not necessary to consider a migration current under these experimental conditions. Hence, the concept may for the present be simplified by regarding the currents measured, as a result solely of the tendency of sulphide ions to move from a region of relatively high to one of low concentration, the rate of this movement being proportional to the concentration gradients. These gradients are then the driving forces.

Part 2. Minimum Collector Concentration vs. Agitation Intensity

Quantitative knowledge of the agitation intensities are thus available for the three conditions of agitation to be employed in this study. Next, with air absent and using a stream of purified hydrogen gas the minimum concentration of collector is determined which suffices to cause adherence of a captive bubble upon the mineral surface using the standard technique.

For intense agitation in the Waring Blendor, the chalcocite specimen is polished and tested in demineralized water to zero angle in the usual manner. It is then overturned and screwed to a ½ in. diam bakelite rod, without removing the specimen from the solution.

The specimen is then transferred to the solution in the Blendor, clamped into the cover assembly previously described, and conditioned for a specified time in 800 cc of collector solution at pH 10.5 with the agitator turned on, and hydrogen flowing at 1 liter per min.

At the end of the conditioning time, the specimen is taken off the rod and immediately placed in the cell used for mild agitation with H_2 gas flowing continuously through the stone diffuser. The solution in this cell has a composition identical with the one in the Blendor. The transfer of the specimen requires less than 30 sec. Between the time when Blendor conditioning ends, and the first bubble contact measurement is made, 1 min has elapsed, allowing 30 sec for bubble contact before measurement.

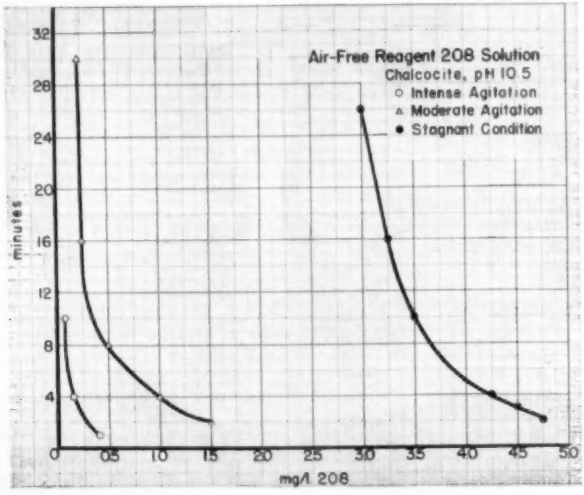


Fig. 7—Critical collector concentration vs. conditioning time.

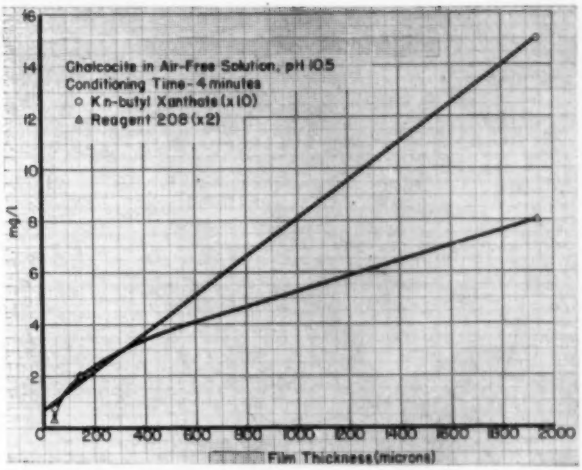

Chalcocite in Air-Free Solution, pH 10.5
Conditioning Time - 4 minutes
○ K n-butyl Xanthate (x 10)
△ Reagent 208 (x 2)
mg/l
Film Thickness (microns)

Fig. 8—Critical collector concentration vs. film thickness.

Measurements on three other areas of the specimen may be made in 2 to 3 min.

Minimum measurable angle with the equipment used was 25°. The curves for this data represent the locus of all points for which any lower concentration produced no measurable angle.

For mild agitation, the specimen, after test for cleanliness as described above with demineralized water, is placed in the cell containing 300 cc lime water at pH 10.5 containing specified collector concentration with H_2 gas through the diffuser and solution at 1 liter per min. This instant represents zero time. Angles are tested minutewise for 6 min, and then every 2 min for a total of 30 min, with H_2 gas flowing continuously at all times to maintain the agitation and oxygen-free solution. The time at which a measurable angle first appears is plotted as the point in time at which the tested collector concentration is a critical minimum value for this degree of agitation.

For stagnant conditioning, this procedure is repeated, except that, after a preliminary purging of the test solution with H_2 gas, the diffuser is held directly above and out of the solution as the specimen is lowered into it.

It is found that the minimum concentration with air absent is a function of the conditioning time as well as the agitation intensity, the relationships being shown in fig. 6 for potassium n-butyl xanthate and in fig. 7 for Reagent 208. Figs. 8 and 9 show how, for a conditioning time of 4 and 10 min, collector concentration is related to the effective fluid film thickness. The relations become linear in the latter indicating degeneracy to a relatively simple process.

The conclusion appears to be inescapable that the process of conditioning involves a heterogeneous surface reaction in which the diffusion of collector across a fluid film barrier is the rate-determining process.

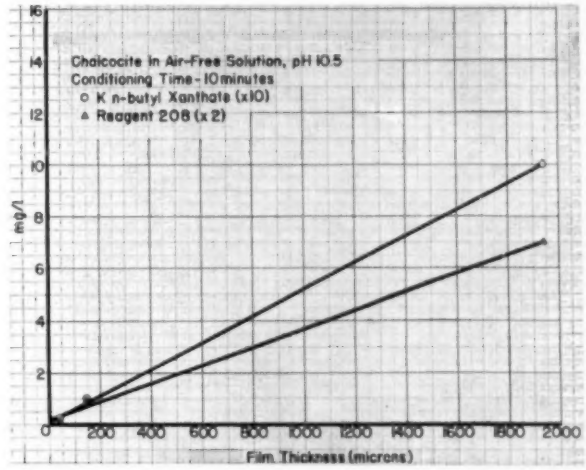

Chalcocite in Air-Free Solution, pH 10.5
Conditioning Time - 10 minutes
○ K n-butyl Xanthate (x 10)
△ Reagent 208 (x 2)
mg/l
Film Thickness (microns)

Fig. 9—Critical collector concentration vs. film thickness.

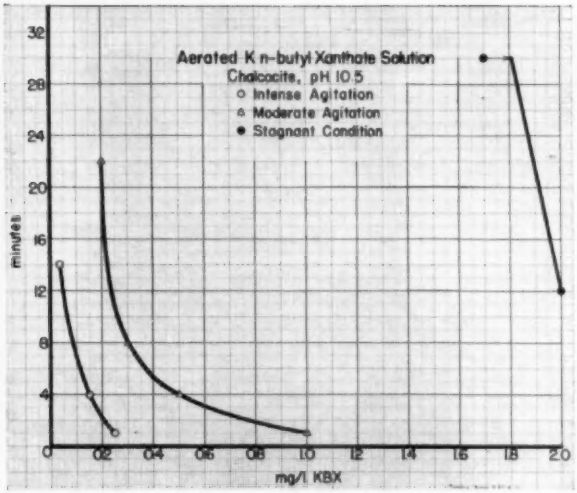


Fig. 10—Critical collector concentration vs. conditioning time.

Part 3. Dissolved Oxygen and Sodium Sulphide

Effect of Oxygen: When the tests described in Part 2 are repeated with air used in place of hydrogen, the figs. 10 and 11 are obtained in place of figs. 6 and 7. The relation of minimum collector concentration and fluid film thickness obtained for 10 min of conditioning, in fig. 9, is changed to that shown in fig. 12.

The comparison shows that a higher collector concentration is required when oxygen is present during conditioning with collector. In other words, oxygen acts as a depressant in this system.

Effect of Sulphide: The most minute amount of dissolved sodium sulphide will effectively depress chalcocite if air is excluded. This may be shown by grinding 30 g of chalcocite in a ball mill with 0.5 g CaO and 200 ml of water, transferring to a 4 liter Fagergren flotation cell, introducing purified nitrogen for 5 min and then adding 150 mg of potassium n-butyl xanthate to produce a strong float, and then 1 ml of a 1.6 pct solution of $Na_2S \cdot 9H_2O$ which depressed the float. It was found that a sample of the pulp solution contained an amount of sulphide which is not detectable with lead nitrate reagent. The same result was obtained in another test using 300 mg of Reagent 208.

Effect of Simultaneous Presence of Sulphide and Oxygen: Figs. 13 and 14 show how sodium sulphide affects the critical concentration of collector when air is present. The method of measurement is identical with that of Part 2. It will be observed that small additions of sodium sulphide reduce the collector requirement while increasing amounts beyond an optimum result in aggravating the requirement. As figs. 13 and 14 illustrate, the optimum sulphide concentration (depending upon the degree

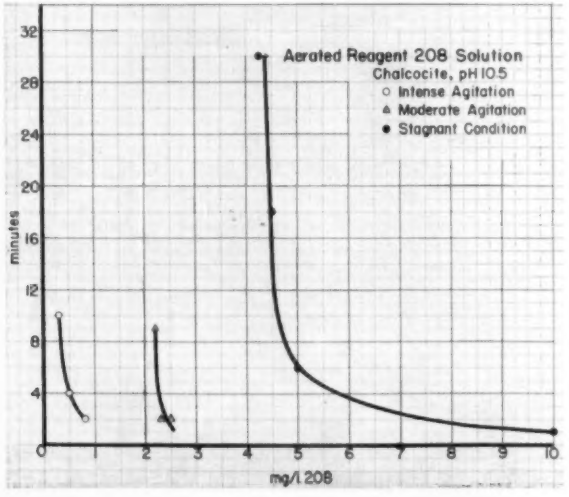


Fig. 11—Critical collector concentration vs. conditioning time.

of agitation) corresponds to a minimum in the initial collector requirement. Comparison in table I shows that these minimum values correspond almost exactly with the minimum values obtained in Part 1, figs. 6 and 7, where sulphide and oxygen are both absent.

Table I. Minimum Critical Collector Concentration

Collector	Agitation	Sulphide and Air Present, Mg per L	Sulphide and Air Absent, Mg per L
K n-butyl xanthate	Stagnant	0.95	0.86
Reagent 208	Stagnant	3.25	3.5
K n-bX	Moderate	0.087	0.08
Reagent 208	Moderate	0.25	0.22

In the region of low concentrations the sulphide appears to act as an activator while at higher concentrations it becomes a depressant. Not only must it be recognized that one reagent may fulfill both roles, depending upon its concentration, but also the question of whether it serves as one or the other depends upon the intensity of agitation. No doubt the system here involving at least three substances in a diffusional process is very complex. As demonstrated later, sulphide serves to react with, and presumably destroys, dissolved oxygen at the interface. It also tends to render the chalcocite extremely reactive electrochemically. The first property may relate to its role as an activator, the latter serve to explain its depressing action. The explanation of

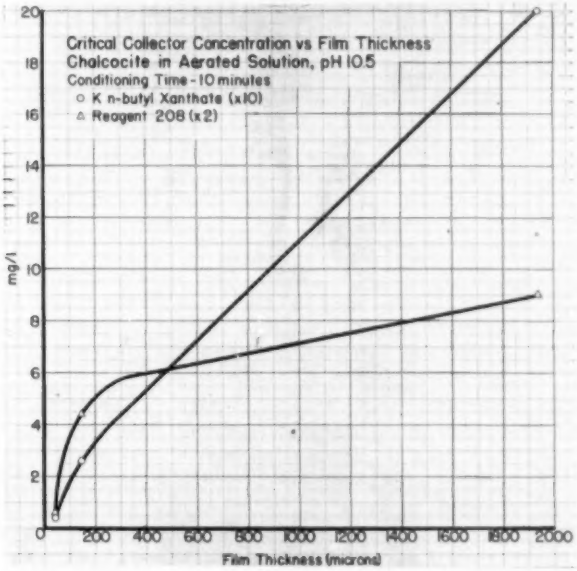

Critical Collector Concentration vs Film Thickness
Chalcocite in Aerated Solution, pH 10.5
Conditioning Time - 10 minutes
○ K n-butyl Xanthate (x10)
△ Reagent 208 (x2)
mg/l
Film Thickness (microns)

Fig. 12—Critical collector concentration vs. film thickness.

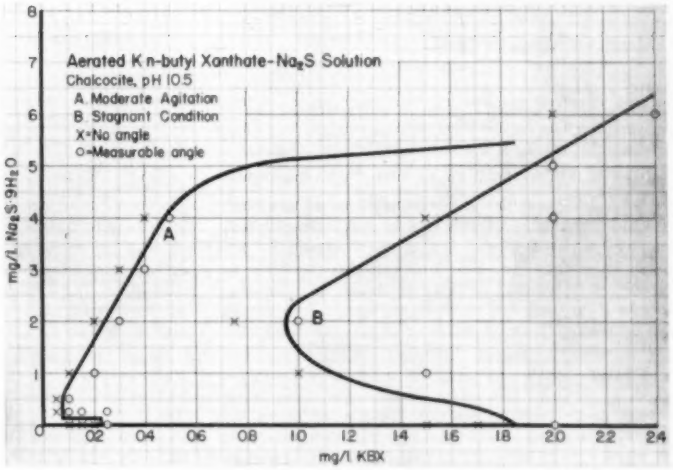


Fig. 13—Effect of sulphide on critical collector concentration.

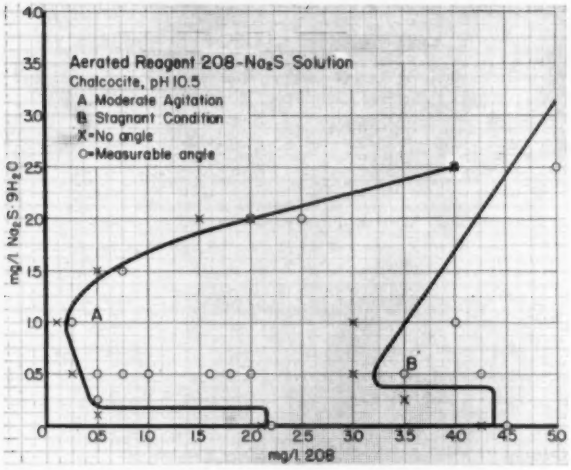


Fig. 14—Effect of sulphide on critical collector concentration.

depression as a surface displacement of all other ions by sulphide is valid at higher concentrations, as a later group of experiments will demonstrate, using sulphide concentrations of higher order.

In connection with the action of sodium sulphide on chalcocite, it is noted that there is a marked increase in reactivity of chalcocite specimens which have been polished and immersed in 0.004 mol sodium sulphide solution pH 10.5 for 1 hr. This is demonstrated by a great increase in sensitivity to the introduction of oxygen to a lime solution containing the specimen and a calomel electrode. The potential in the absence of oxygen is approximately the same as that of a freshly ground surface during and immediately after abrasion on a carborundum wheel (about —400 mv). On the other hand, a chalcocite surface which has been exposed for some length of time to the air, becomes quite insensitive; its potential is about —80 mv. The sensitized chalcocite is also capable of displaying great reactivity to sulphide; this is demonstrated by the current-potential curves for sulphide electrochemical oxidation given above, which are curves for a chalcocite specimen conditioned in sulphide for 1 hr.

It is important to recognize that amounts of sodium sulphide much smaller than determinable by ordinary analytical methods nevertheless have a profound effect upon flotation of chalcocite. This is of interest in view of the observations that sulphide ion is reported to be present in some pulps.[5]

Part 4. Electrochemistry of Sulphide and Oxygen

For purposes of analysis, it may be assumed that oxidation of sulphide at the mineral-liquid interface is electrochemical. If it is assumed to be electrochemical it does not matter whether or not the anode and cathode reactions occur separately upon discrete areas. It will help to postulate that the function of oxygen may be assumed by a positive electric current, that of sulphide by a negative electric current. Thus, the effects produced by the two reactants may be studied individually and quantitatively by means of electrical measurements.

If the sulphide solution is saturated with air, the cathodic reaction is not limited by diffusion inasmuch as oxygen is relatively plentiful. Nevertheless, in concentrations as high as 52.8 mg per liter of $Na_2S \cdot 9H_2O$ the potential in volts of a chalcocite surface under "moderate" agitation is almost the same in an air-saturated solution as it is in one purged with nitrogen. This means that in this range it is the sulphide which exerts the dominating influence in determining the surface properties.

Imposing a positive current upon such a surface moves the potential in the positive direction in the air-saturated solution containing sulphide. As shown in fig. 15*a* for the same agitation condition, namely, moderate, mean fluid film thickness 147 microns, it will be seen that at first the current for any given high negative potential rises as the potential is increased whether the oxygen is present or not. Moving along in the direction of increased potential, the differences between the two curves begin to augment. Finally, a point is reached at which the cathodic activity builds up to a point at which the current density to maintain the potential shifts suddenly from a positive to a strongly negative value. Whereas heretofore it was required to supply positive current to the chalcocite it is now necessary to furnish negative current. The curve then continues as shown in fig. 15, crossing the x-axis again at —0.155 v in the case of 52.8 mg per liter $Na_2S \cdot 9H_2O$.

In the absence of an imposed current the mineral surface assumes the potential for zero current, that is, the intersection of the graph with the x-axis. However, it is evident that under these conditions of study, two such potentials are possible. On the one hand, a stable potential for zero current represented by the left-hand intersection in the graph, corresponds to the presence of unoxidized sulphide ion at the surface. On the other hand, the right-hand intersection is interpreted as a stable potential corresponding to excess oxygen at the surface. A transition from excess sulphide to excess oxygen at the surface, in this high concentration of sodium sulphide, requires the addition of energy to the system. In this instance, this is accomplished by im-

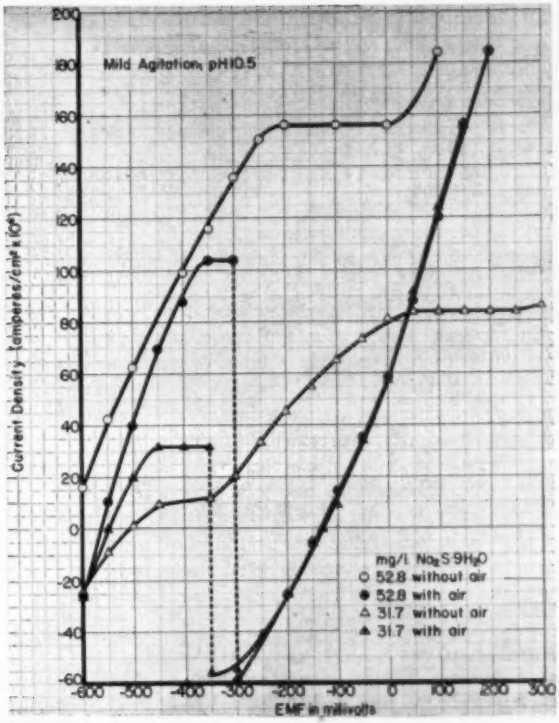
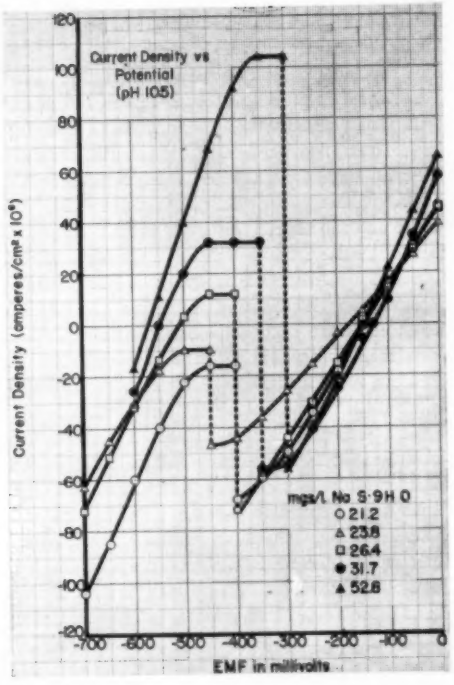


Fig. 15a (right)—Chalcocite in sulphide solution.

Fig. 15b (below)—Chalcocite in sulphide-air solution.



posing a positive current, which has the effect of depleting the sulphide ion concentration at the surface. The analogous transition from excess oxygen to excess sulphide at the surface is accomplished by depleting the oxygen concentration at the surface; namely, by purging the solution with nitrogen, or, by allowing the surface to be conditioned for a considerable time in a stronger sodium sulphide solution.

A characteristic curve is obtained for each concentration of sodium sulphide used. Those for the lower concentrations reveal a transition from excess sulphide to excess oxygen lying wholly below the x-axis. This is taken to mean that the sulphide depletion required to effect the transition is brought about by means of the added oxygen alone. From the assumptions made herein, this is accomplished by virtue of the cathodic activity of oxygen when it reaches the mineral surface. On this basis, the range of potentials covered by the transitions comprise a region in which oxygen exhibits strong cathodic activity, presumably to form hydroxyl ions. In this system, the extent to which this activity occurs is dictated by the requirement of sulphide depletion at the surface; when the sulphide has reduced very nearly to zero concentration at the surface, the oxygen activity would then have reached a maximum rate. This view is supported by comparison of curves in fig. 15a. The transitions occur at potentials somewhat lower than those corresponding to the upper limit of the sulphide diffusion currents under hydrogen. This suggests that the

condition for a transition from excess sulphide to excess oxygen is a concentration of sulphide ion approaching zero at the interface. Admittedly, a more satisfactory substantiation of this inference would be preferred.

The sulphide concentration corresponding to the maximum which oxygen alone can deplete spontaneously in this system of agitation intensity lies, as shown in fig. 15, between 23.8 and 26.4 mg per liter of $Na_2S \cdot 9H_2O$. Fig. 16 shows contact angles measured upon a specimen of chalcocite conditioned prior to each test in stronger sodium sulphide and then transferred to the solution containing the specified concentration of sodium sulphide with an amount of xanthate or thiophosphate collector as indicated.

The contact angles of fig. 16 are those measured 1 min after the specimen enters the solution. This is the shortest time experimentally convenient. Over longer periods of time the contact angle grows less and ultimately disappears.

A critical sulphide concentration of 24.5 to 26.5 mg per liter from fig. 16 compares well with 23.8 to 26.4 mg per liter from fig. 14. This concentration is one below which a contact angle is obtained; it does not necessarily persist.

All of the above refers to a system saturated with air and under "moderate" agitation. On the other hand, if nitrogen or hydrogen is used to purge the solution, the merest trace of sulphide precludes a contact angle.

Experimental Procedure: *Preparation of Specimen:* A piece of synthetic chalcocite, made as previously described, was mounted in lucite and polished with levigated alumina in the conventional manner. The exposed polished area was 1.25 sq cm. It was then treated with 0.004 molar sodium sulphide in lime solution at pH 10.5 for 1 hr.

The specimen was then transferred with care to remove all liquid, but without washing, to a glass contact-angle cell containing 300 cc of a solution of lime to bring it to pH 10.5 after addition of collector and sodium sulphide.

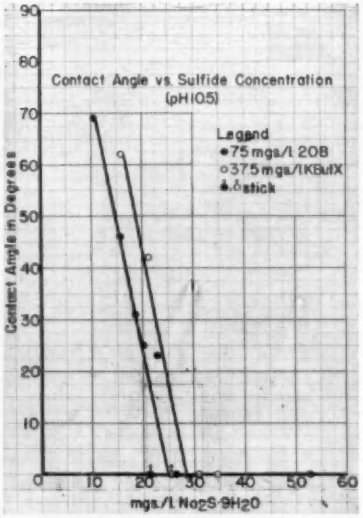


Fig. 16—Chalcocite in sulphide-air collector solution.

Contact Angle Measurements: When the prepared specimen is placed in the cell a stone diffuser is introduced through which air passes at the rate of 1 liter per min. The contact angle measurement is made within the first minute after the specimen is placed in position. We have observed that these contact angle values can be obtained only when the sulphide and collector concentrations are not varied during the preparation for measurement, namely, by washing into distilled water for angle measurement. Angle values obtained in the specified media

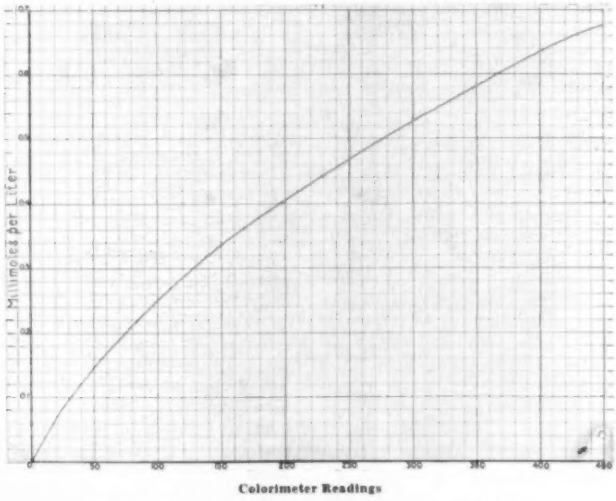


Fig. 17—Light absorption vs. concentration of Na_2S.

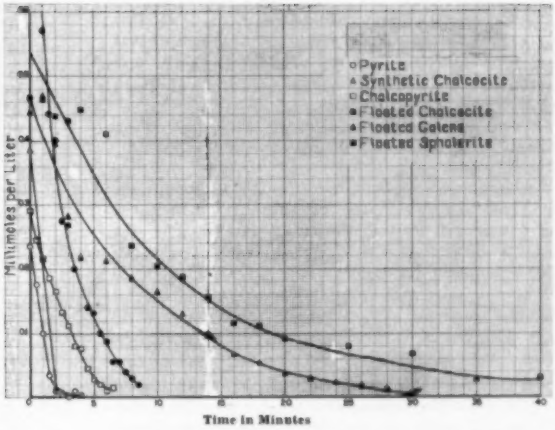


Fig. 18—Na_2S disappearance, minerals in air.

remain during the first minute, but tend to reduce to zero in the next few minutes at a rate which depends upon the sulphide concentration.

Current-potential Measurements: The equipment and procedure has been described in Part I. When the concentration of sodium sulphide is less than the critical value, the solution is purged with nitrogen for a few minutes, during which period the potential is observed to fall rapidly. When a minimal negative current necessary to maintain say —600 mv is attained, the air is admitted in place of nitrogen. The current changes only slightly, and a position of stability is soon reached for each succeeding potential setting on the curve. With sulphide in excess of the critical value no preliminary nitrogen treatment is necessary to cause the potential to fall readily to a stable negative value.

Part 5. Oxidation of Sulphide in Pulps

An intensely active reaction between dissolved sulphide and oxygen upon a suitably conditioned chalcocite surface has been made evident from the preceding electrochemical study. As a consequence one would expect that sulphide would be rapidly consumed in flotation pulps containing sulphide minerals such as chalcocite. It will now be shown that this is indeed the case.

Experimental Procedure: A Denver 8 x 8 in. laboratory ball mill was used to grind 30 g samples of various minerals with 0.5 g of c.p. lime and 200 cc of water. Grinding time was 5 min. The charge was transferred along with all washings to a Fagergren laboratory flotation cell of 4 liter capacity. Sulphide was added in an amount needed to satisfy any absorption by the mineral and leave a measurable excess. The machine agitator was run in the usual manner for flotation testing and pulp samples were withdrawn at intervals for determination of their soluble sulphide content. In those tests for sulphide disappearance anaerobically the flotation cell was fitted with a tightly drawn pliofilm cover. Nitrogen (water pump) purified with alkaline pyrogallol was passed into the air inlet cock of the flotation cell. In these tests samples were withdrawn through the bottom opening of the cell.

Temperature of pulps was 14 to 18°C (57.2 to 64.4°F).

Synthetic chalcocite was made by precipitation of cupric sulphide which was then melted. A solution of 9 lb of copper sulphate (commercial blue vitriol) was dissolved in 9 liter of water. Hydrogen sulphide was passed into this solution until precipitation of cupric sulphide was complete. After decanting several times with water, the material was filtered, oven-dried, and melted in a graphite crucible heated by induction. The melt was allowed to cool under a cover of sulphur, after which the crucible was broken, leaving a sound casting, the analysis of which is given in table II.

It was not possible to obtain chalcocite sufficiently free of gangue and pyrite inclusions, and therefore synthetic as well as the natural mineral was used.

Analytical Procedure: A sample of pulp was drawn and at once filtered through a small Buchner funnel until at least 50 cc of filtrate was obtained. Exactly 50 cc of this pulp liquor was mixed with 50 cc of a solution containing 0.2 pct glacial acetic acid and 0.02 pct lead nitrate. A brown color developed immediately. After standing for 2 min the light absorption of this colloidal solution was measured with a Klett-Summerson photoelectric colorimeter. From the previously obtained calibration curve for this instrument obtained with known additions of sodium sulphide, see fig. 17, the sulphide concentration in the pulp liquor was obtained. The sulphide solution used for calibration purposes was itself standardized by titration with standard iodine and thiosulphate, having been prepared from the best obtainable grade of sodium sulphide ($Na_2S \cdot 9H_2O$) crystals.

Results: The time rate of disappearance of the sulphide from the pulps is presented in the curves of figs. 18 and 19 and in table IV. In a blank run made, using all of the reagents and conditions including ball mill grinding, but omitting the addition of any mineral, the sulphide concentration dropped at a very slow rate which was, however, appreciably greater than when lime and plain tap water were used without grinding. In the latter case the sulphide concentration appeared unchanged after more than 30 min.

Pyrite, chalcopyrite and chalcocite, both natural and synthetic, caused a disappearance of sulphide at a rate which would be considered rapid when related to time ordinarily allowed for conditioning and flotation in mill circuits.

Galena and sphalerite, though contributing appreciably to sulphide disappearance, do so at a relatively low rate.

Not only the mineral, but the air is essential to the rapid disappearance of sulphide with pyrite and chalcocite. This is shown in fig. 19 comparing the rate in nitrogen with that in air, other conditions remaining the same.

It is also apparent from the results of these tests that the minerals chalcocite and pyrite, which are most effective in the disappearance of sulphide, can repeat their performance substantially unimpaired when a fresh addition of sulphide is made following the complete disappearance of a previous addition. In this sense the process of sulphide disappearance may be said to be catalytic. This is not intended to imply that the role of the mineral, or of any product thereof, is that of a true catalyst inasmuch as this study does not afford any evidence to substantiate such a hypothesis. It is merely that from a practical standpoint the effect insofar as flotation is concerned is equivalent to a catalytic oxidation.

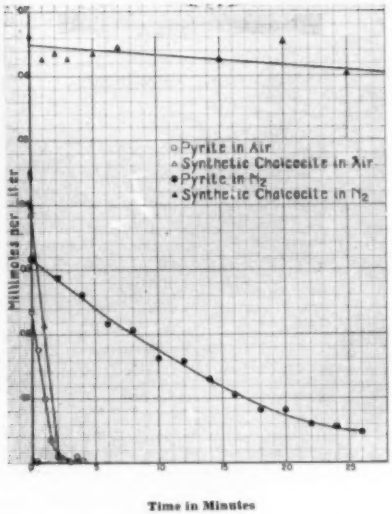


Fig. 19—Na_2S disappearance in air and in N_2.

It is to be understood that all reference herein made to sulphides does not necessarily apply to complex soluble sulphides such as polysulphides.

Table II. Composition of Minerals

Mineral	Cu	Fe	Pb	Ni	Zn	S	SiO_2	CaO	MgO	Al_2O_3
Pyrite	0.09	47.30	Trace		Nil	51.61	0.78			
Chalcopyrite	28.70	30.63	0.22	Trace	Trace	34.48	2.75	0.68	0.26	1.05
Chalcocite floated	35.05	4.22	0.46	Nil	1.07	11.62	45.31	0.80	0.19	0.64
Galena	Nil	0.16	84.81	Nil	0.11	13.35	0.70	Nil	0.21	Nil
Sphalerite		3.21	0.28		63.35	32.76	0.47			
Synthetic chalcocite	78.40	0.32		0.06		20.04	0.21			

Table III. Size Analysis of Minerals

Mineral	Galena			Pyrite			Sphalerite			Synthetic Chalcocite		
Mesh	Wt.	Pct	Cum. Pct	Wt.	Pct	Cum. Pct	Wt.	Pct	Cum. Pct	Wt.	Pct	Cum. Pct
On 65	0.02	0.02	0.02	0.4	0.4	0.4	0.3	0.3	0.3	1.8	1.8	1.8
100	0.03	0.03	0.05	1.8	2.0	2.4	1.4	1.4	1.7	5.5	5.6	7.4
200	0.20	0.21	0.26	12.8	14.0	16.4	8.9	8.7	10.4	25.6	26.1	33.5
270	0.33	0.34	0.60	9.5	10.4	26.8	7.6	7.5	17.9	13.9	14.2	47.7
325	0.19	0.20	0.80	5.3	5.8	32.6	4.5	4.4	22.3	6.7	6.8	54.5
400	17.3	18.0	18.8	22.9	25.0	57.6	16.8	16.5	38.8	26.7	27.2	81.7
500	16.9	17.6	36.4	6.4	7.0	64.6	18.1	17.8	56.6	11.6	11.8	93.5
800	14.3	14.9	51.3	8.5	9.3	73.9	11.0	10.8	67.4	2.5	2.5	96.0
1120	10.2	10.6	61.9	5.0	5.5	79.4	7.4	7.3	74.7	0.4	0.4	96.4
1600	7.0	7.3	69.2	2.7	3.0	82.4	4.2	4.1	78.8	0.1	0.1	96.5
Through 1600	26.5	30.7	99.9	16.2	17.7	100.1	21.7	21.3	100.1	3.4	3.5	100.0

Table IV. Sulphide Disappearance Rate for Successive Additions of 5 Pct $Na_2S \cdot 9H_2O$[a]

Time, Min	Na_2S Millimols per Liter					Time, Min	Na_2S Millimols per Liter			
0	0.08	0.10	0.115	0.136	0.145	0	0.37	0.385	0.445	0.608[b]
1	0.065	0.045	0.064	0.024	0.050	1		0.194	0.174	0.604
2	0.02	0.003	0.009	0.005	0.024	2	0.010	0.012	0.010	0.600
3	0.01	0.005	0.008	0.020	0.030	3				
4			0.002			4				0.324
5				0.090	0.003	5				
6						6				0.012

[a] 10 cc 5 pct $Na_2S \cdot 9H_2O$ = 2.08 millimols Na_2S [b] 40 cc added at one time

Acknowledgment

George Bobadilla and Mrs. Dolores Casella performed the experimental work upon which this paper is based.

Thanks are due to Phelps Dodge Corp. and to W. H. Osborn, Director of its Department of Development and Research, for permission to publish this work.

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Canadian Deposits of Uranium and Thorium

by W. F. James,
A. H. Lang,
Richard Murphy
and
S. N. Kesten

The paper is a summary of information on Canadian deposits of uranium and thorium up to the end of 1948, prepared by geologists of Eldorado Mining and Refining Ltd. and by the specialist on uranium deposits for the Geological Survey of Canada. The paper contains a history of Canadian uranium and thorium discoveries, a working classification of the deposits, fairly detailed descriptions of the Eldorado mine and of properties in northern Saskatchewan, and shorter accounts of several other discoveries.

Introduction—by W. F. James and A. H. Lang

This paper describes the geology and present state of development of uranium and thorium deposits in Canada. It is expected that this information will be added to that now being assembled from available American and British data to provide a background for engineers and others who may be called upon in the future to examine or search for radioactive deposits. Some generalizations regarding the Canadian deposits are included, but many of these are necessarily tentative. The information contained in the paper is up to date for the end of 1948.

W. F. JAMES is with Eldorado Mining and Refining (1944) Ltd., A. H. LANG is with the Geological Survey of Canada, RICHARD MURPHY, Member AIME, is with the Newmont Mining Corp. of Canada, and S. N. KESTEN is Field Geologist, Eldorado Mining and Refining (1944) Ltd.

AIME San Francisco Meeting, February 1950.

TP 2769 I. Discussion (2 copies) may be sent to Transactions *AIME before March 30, 1950. Manuscript received May 9, 1949.*

The information contained in this paper has been derived from many sources. Much of it is from the records of the Canadian government-owned Eldorado Mining and Refining (1944) Limited and from examinations of government and private properties made by officers of the Geological Survey of Canada. Some information on privately-owned properties has been quoted from other sources mentioned in the text.

The information has been assembled by W. F. James of Eldorado Mining and Refining (1944) Limited and A. H. Lang of the Geological Survey of Canada, who have written the sections for which they are credited. The description of the Eldorado mine is based on reports by Richard Murphy, formerly chief geologist for Eldorado; the report by Mr. Murphy was published previously by the Canadian Institute of Mining and Metallurgy,[1] to whom the writers are indebted for permission to republish. The section dealing with the geology of and development work in the Goldfields area, where Eldorado has important holdings, is by S. N. Kesten, field geologist for Eldorado. The geological diagram of the Goldfields area (fig. 2) was kindly supplied by A. M. Christie of the Geological Survey of Can-

ada, and is a much simplified version of his fairly detailed mapping of the region.

The paper includes a brief history of the most significant discoveries of radioactive minerals in Canada and a statement of the controls exercised by the Federal government. It discusses the known types of Canadian uranium and thorium deposits and provides a working classification. Several deposits are then described briefly, in the order of the classification.

History

The first Canadian uranium discovery was made at Lake Superior and described in 1847 by J. L. LeConte, a distinguished American geologist. Several attempts to rediscover this occurrence were unsuccessful because the location was described indefinitely. A discovery known as the Camray, made in 1948 by a prospector searching for the old occurrence, has attracted wide attention and aroused considerable interest in the circumstances surrounding the early find. Therefore it may be of interest to describe them fairly completely.

LeConte stated[2] that he had received a specimen from the Lake Superior region, and continued:

> This mineral forms part of a collection made by Mr. B. A. Stanard on the north shore of Lake Superior .... This mineral as I am informed by Mr. Stanard, occurs on the north shore of Lake Superior, about 70 miles from the Sault Ste. Marie, at the junction of trap and Sienite; the vein in which it is found is about two inches in width; but on account of its position (on the face of an almost perpendicular cliff) only a few specimens were obtained, and those with great difficulty.

The Mr. Stanard referred to was probably Benjamin Stanard, who was captain of a schooner operated on Lake Superior at that time by the American Fur Company.[3]

LeConte considered that the mineral was closely related to pitchblende, but because of slight physical and chemical differences between it and the typical pitchblende of Joachimsthal in Bohemia, he classed it as a new mineral which he called "coracite." He probably derived that name from the Greek word for "raven," in allusion to the black color of the mineral.[4]

J. D. Whitney published a note on the mineral in 1849.[5] He believed that it contained uranium of a different valence from that of typical pitchblende, and that it was more readily soluble in acids.

In 1857 F. A. Genth published the following description of the Stanard specimen:[6]

> Dr. John L. LeConte kindly presented me with a specimen of the mineral from about 90 miles above Sault Ste. Marie on the north shore of Lake Superior, which he had described as coracite. Its great resemblance to pitchblende favoured the opinion that it was really nothing else ... though it is interesting that it is so readily soluble in chlorhydric acid, this fact alone is not sufficient to separate it from pitchblende.

The first geological studies of the region were made by the Geological Survey and described by W. E. Logan in "The Geology of Canada," published in 1863. Under the heading "Uranium," the report states "An ore of this rare metal is said to occur at Mamainse," then continues with a condensation of the data published by LeConte and Whitney.[7]

The occurrence was summarized in four later reports of the Geological Survey, and officers of the survey made inquiries at various times in efforts to relocate the discovery, and answered numerous requests from prospectors for information on the occurrence. All efforts were unsuccessful, however, because of the conflicting locations given in the different publications. It now seems likely that LeConte's estimate of 70 miles from Sault Ste. Marie was nearly correct and that Genth's mention of 90 miles was unfortunate. It also appears likely that the use of the name Mamainse in "The Geology of Canada" followed an old custom of referring to the entire region as Mamainse, whereas more recent investigators have assumed that it referred to Mamainse Point which is considerably south of the recent discovery.

Robert Campbell, discoverer of the Camray deposit, stated that during the winter of 1947 to 1948 he studied all the references to the old coracite occurrence. He formed the Camray Prospecting Syndicate with a capital of $4000. subscribed by Toronto backers, and spent the season of 1948 prospecting by boat and on foot along the shore of Lake Superior in the general region of the old discovery. During the earlier part of the season he had two assistants, but when they left he continued his search alone. He was equipped with a Geiger-Müller counter. On September 8, he obtained a strong reaction on his counter near the south wall of a gorge on Theano Point, close to the water's edge, and on close inspection found that the reaction came from a stringer of black mineral. Other stringers, some of which contained uranium stain, were found at intervals along the wall, over a total distance of about 600 ft. He staked 30 claims and recorded them on October 16.

The discovery agrees so closely with LeConte's description that it is considered a rediscovery. News of the find caused the greatest staking rush that has occurred in Ontario for over 20 years. Over 2000 claims were staked in a few months and other discoveries were reported, but snow has prevented examination of these up to the time of writing.

After 1904, radioactive minerals were identified in numerous specimens submitted from many parts of Canada. Practically all were from pegmatite deposits, but one occurrence in Ontario[8] and one in Quebec[9] consisted of small veins of uranium-bearing hydrocarbon related to anthraxolite. None of these occurrences has been developed commercially, but considerable work has been done on pegmatite deposits in the Wilberforce region of southern Ontario with a view to producing uranium as a by-product of mining for fluorite and other minerals.

In 1914, interest in radium caused the governments of British Columbia and Ontario to offer bonuses for discoveries of radioactive minerals in commercial quantities.[10] The rewards were not claimed and the offers were withdrawn.

The outstanding date connected with Canadian radioactive discoveries is 1930, when Gilbert Labine and E. C. St. Paul found pitchblende, cobalt, and native silver in veins at Great Bear Lake, just south of the Arctic Circle. The Eldorado Gold Mining Co. was organized to operate the mine and a radium refinery was built at Port Hope, Ontario. Great dif-

ficulties of transportation, climate, and treatment processes had to be overcome before production was begun in 1933. In 1940, difficulties of wartime operation caused a suspension of mining. During the years between the discovery of the Eldorado mine and the beginning of the last war several other pitchblende discoveries were made in the Great Bear Lake region, and one of these properties, now called International Uranium, produced a relatively small amount of pitchblende concentrate incidental to mining native silver. In 1935, pitchblende was identified in samples from the Nicholson gold-copper prospect at Lake Athabaska, but further development did not take place until recently.

When atomic fission was perfected during the war the Canadian deposits, previously of interest as sources of radium, assumed outstanding importance as British-American sources of uranium. It was decided, accordingly, that they should be placed under state control. This was done by placing a temporary ban on staking and mining for, and trading in, radioactive substances, and by purchasing the shares of the Eldorado company at an agreed price and transferring them to a "Crown company" operated by the Federal Government. The Eldorado mine was reopened in 1942 with as little publicity as possible. Equipment was added to the concentrator, and the refining method was changed from a batch process to one of continuous extraction with emphasis on uranium production. The Crown company established its own air service and bought and expanded the barge transportation system that serves the Mackenzie District. In addition to servicing the Eldorado mine which is 1350 miles by rivers and lakes from railhead, it supplies the private freighting requirements of the Mackenzie District, other than those of the Hudson's Bay Co.

The Crown company engaged in prospecting and geological studies, with the aid of the Geological Survey of Canada. Several discoveries were made, staked, and tested. Emphasis has been placed on those near Lake Athabaska, which are only one-quarter as far from railhead as is the Eldorado mine, and where climatic conditions are more favorable.

A few years after the war, the Atomic Energy Control Board decided to permit private staking and mining for radioactive substances in order to stimulate uranium discoveries and production, but to retain the Eldorado company as a government-owned producer and as the only refining and marketing agency. Revised regulations were issued,

Fig. 1—Main geological features of Canada, and location of properties and districts mentioned in text.

1—Hazelton View. 2—Gem. 3—Eldorado Mine. 4—Hottah Lake. 5—Giauque. 6—McLean Bay. 7—Goldfields. 8—Black Lake. 9—Lac la Ronge. 10—Camray. 11—Wilberforce. 12—Otter Rapids (thorium).

therefore, in March 1948, and subsequent amendments have been made. Regulations pertaining to prospecting and development work are now essentially as follows:

1. Discoverers must notify the Atomic Energy Control Board or the Geological Survey of Canada giving the location and other details as soon as possible after there has been reasonable opportunity to stake, and before making any public announcement. Such information is treated confidentially.

2. Permits must be obtained from the Board before proceeding with any development work other than that required for making a discovery. This regulation is not made as a deterrent to development work, but so that the Board will always be aware of what is being done.

3. Permits must be obtained from the Board before removing any bulk samples or other shipments of radioactive material, except that specimens for display or ordinary samples for assay may be removed without an order.

The government guaranteed to buy, for five years, all acceptable ores and concentrates with a minimum uranium content equivalent to 10 pct by weight of uranium oxide, at a floor price of $2.75 per pound of contained uranium oxide. The price is f.o.b. rail at any point in Canada, the government absorbing the losses and refining cost. This offer was later extended for another two years, to March 31, 1955.

The attitude of the Canadian Government now is to encourage in every way prospecting for uranium minerals by mining companies and individuals. It is recognized that, initially at least, considerable technical help will be needed, and this is given principally by the Geological Survey, the Bureau of Mines, and several Provincial Mining Departments. It is felt that a prospector might hope ultimately to acquire the same compensation from a discovery of a uranium deposit as he would from the discovery of a gold prospect of comparable size and grade. As a result of the government action, interest among prospectors was aroused and throughout the past season, prospecting has been active in many of the Provinces of Canada, principally in Ontario, Saskatchewan, and British Columbia, and in the Federally administered Northwest Territories. Although to date none of the recently discovered properties has reached the production stage, it is expected that eventually a number of properties will produce significant amounts of uranium concentrates. In the meantime, prospecting by Eldorado has continued. Underground operations have been undertaken on one new Eldorado property and are planned on several others. Publicity regarding discoveries and properties under development is permitted, but production and ore-reserve figures are not allowed to be published.

Types of Canadian Deposits

Uranium and thorium occur in Canada in deposits of several different types. Fig. 1 shows the main geological divisions of Canada and the location of properties and districts. All of these types are listed below, for the sake of completeness, but only pitchblende deposits believed to be of hydrothermal origin are of known economic importance, at least at present.

Uranium deposits:

1. Original constituents of granitic rocks.
2. Pegmatite deposits.
3. Hydrothermal deposits
   - A. Veins, stringer systems, and disseminations
   - B. Giant quartz veins.
4. Bedded deposits in sedimentary strata.
5. Deposits of secondary uranium minerals in gossans, etc.

Thorium deposits:

1. Pegmatite deposits.
2. Hydrothermal deposits.
3. Bedded deposits in sedimentary strata.
4. Placer deposits.

Most of the discoveries are in the Canadian pre-Cambrian Shield. A few mineral occurrences have been reported from the Appalachian Region, and some discoveries that may prove to be important have been made recently in the Cordilleran Region.

The present uranium discoveries in the Shield are strikingly restricted to its western and southern margins. This may be partly due to accessibility, as most of the discoveries have been made near the large lakes that flank the Shield, and more complete prospecting of other regions may alter the pattern of discoveries. On the other hand, the distribution may be explained more theoretically by the facts that many of the age determinations on pitchblende from the Shield indicate Proterozoic and early Paleozoic ages, and that these deposits occur in belts of Proterozoic mountain building. This suggests that other areas of Proterozoic orogeny may be worth prospecting.

Pitchblende-bearing veins and stringers occur in many types of sedimentary, volcanic, and granitic rocks. Therefore no sweeping generalizations regarding favorable host rocks can be made, but some generalizations can be made for individual districts. Many of the deposits in the northwestern part of the Shield occur in or alongside prominent northeasterly-trending fault zones. The individual ore shoots constituting a deposit are generally rather small, and they tend to occur in tension fractures related to the fault zones.

The minerals with which pitchblende is most frequently associated are cobalt-nickel minerals, native silver, and hematite. It is convenient to divide the pitchblende deposits, other than the giant quartz-vein type, into two general classes, one containing complex mineral assemblages and the other containing chiefly pitchblende, hematite, carbonate minerals, and quartz. The writers consider the Camray occurrence a special example of the latter class, in which no hematite has as yet been found.

Most of the other types of deposits are discussed briefly in later sections of this paper.

Granite and Pegmatite Deposits

Many samples of granitic rocks have been shown to contain about 0.003 pct uranium oxide equiva-

Fig. 2—Generalized geological map of Goldfields region.

(Map by A. M. Christie, Geological Survey of Canada.)

lent, and contents up to 0.01 pct are not uncommon (H. V. Ellsworth, personal communication). All information available at present indicates that the uranium in these rocks occurs in original rock-forming minerals, and has not been introduced later.

Minor quantities of uraninite and, to a lesser extent, of other uranium minerals, have been found in numerous pegmatite bodies associated with granitic rocks. Such deposits occur in widely separated parts of Canada, but particularly in the southern part of the Canadian Shield, in a belt extending from Lake Huron to the Gulf of St. Lawrence. None of these has yet been proved to be a commercial source of uranium." The most promising pegmatite deposits seem to be those containing other valuable minerals such as feldspar, fluorite, mica, apatite, beryl, and lithium minerals, from which uraninite might be recovered as a byproduct. Other uranium-bearing pegmatite deposits were found at Lac la Ronge, Saskatchewan, in 1947 and 1948. In view of the large number of uranium-bearing pegmatites already known, prospecting for others cannot be recommended highly until such time as some of the known occurrences are shown to be productive.

Considerable thought has been given to the question of whether areas containing uranium-bearing pegmatite deposits, or granites with higher-than-

average uranium content, would be favorable localities for the discovery of greater concentrations in the form of hydrothermal deposits. It seems quite probable that such granitic rocks would have provided vein deposits but, theoretically, the hydrothermal solutions would be expected to have migrated farther from their source than did the pegmatite-forming material. Consequently, hydrothermal deposits, derived from the same sources as the granite or pegmatite, would not be expected to occur in the immediate vicinity of the pegmatite, but in the border zones of the intrusions, and many such deposits may have been removed by erosion. The pitchblende deposits that do occur in granite seem to be much later than the granite, and are believed to have been derived from deep-seated sources younger than and unrelated to the host rocks of the deposits.

The Eldorado Mine—by Richard Murphy

The Eldorado mine (fig. 1) is at Labine Point on Great Bear Lake, 28 miles south of the Arctic Circle. The region is one of sparse vegetation, and ice remains in the lakes for all but a few summer months. The general geology of the area was described by D. F. Kidd of the Geological Survey,[12] and later work was done at and near the mine by other officers of the Survey and by the writer.

General Geology: The mine is in the pre-Cambrian Shield and all the rocks nearby are believed to be of Proterozoic age.[13] They may be tabulated as follows, with the oldest rocks at the bottom of the table:

Intrusives
9. Late diabase
8. Early diabase
7. Aplite
6. Granite
5. Diorite

Echo Bay Group
4. Porphyritic lavas and pyroclastics
3. Feldspar porphyry
2. Sediments
1. Massive tuff

At LaBine Point the principal rocks are the lowest recognized members of the Echo Bay group. At the base of the series is a fine to dense, dark uniform rock of uncertain origin, but presumed to be a tuff. Very little is known as to its relation with the succeeding sedimentary member, which is highly quartzose and varies widely in appearance. It includes fine- and coarse-banded quartzites, dense cherts brecciated by irregular chloritic seams, and vaguely banded metamorphosed silicic rocks. These are buff to pink in color, much stained with hematite. Magnetite is a common constituent as disseminations and veinlets and replaces certain bedded zones to a degree that suggests iron formation. Thinly bedded argillaceous horizons are traceable for a few hundred feet on surface and underground, but for the most part the different phases of the quartzite are distributed in no regular manner. There are very minor amounts of limestone and conglomerate.

The feldspar porphyry is a fine-grained crystalline rock with small scattered phenocrysts. Where freshest, the rock is purplish-gray and the phenocrysts are light in color and prominent. With alteration the rock is generally redder, in extreme cases grading into a dense, reddish brown phase. In least altered areas the porphyry is uniform, but through the zones of alteration, it is very irregular in character. Its identity is rarely in doubt, however, because there are usually recognizable remnants.

The porphyry forms bands and masses in the sediments and has not been found in other members of the Echo Bay group. In general, the bodies of porphyry conform roughly to the structure in the sediments and could, therefore, be either sills or flows. Since the contacts are obscure, and are rarely placed with certainty within 5 ft, the details are not helpful. On Cobalt Island, the porphyry sends several stubby apophyses into the sediments though the contact is elsewhere strictly conformable. There are a few instances in which masses of porphyry appear to truncate the sediments at low angles. At one locality, now concealed by tailings, several blocks of sediments were found as inclusions in the porphyry up to a distance of 20 ft from the contact. While not conclusive, the evidence in the mine area is that the porphyry is an intrusive.

A little further afield structures anomalous to an intrusive are found in the porphyry. Exposures show faint brecciation and occasional banding, and in one instance well defined pillows. Breccia commonly appears near the upper part of the porphyry band and might be interpreted as a fragmental zone topping a flow. Since there are other explanations for these relatively obscure structures, they are not considered to be proof that the porphyry is not an intrusive, but merely serve to keep the question open.

In contrast with the sediments, the overlying lavas and pyroclastics are fresh in appearance. The flows are porphyritic andesites with a dense, gray groundmass, and prominent feldspar and hornblende phenocrysts. In certain horizons amygdules are abundant. The interbedded pyroclastics consist of breccia, agglomerate, and thinly-bedded tuff. The volcanics invariably weather brownish as opposed to the decidedly reddish color of the underlying formations. Higher up in the series conglomerate and sandstone are found with the flows, but no important amounts occur near the mine.

The oldest major intrusive is diorite, a medium to coarse grained rock, showing considerable variation. The diorite occurs some two miles northeast of the mine as a large elongate body, cutting across the Echo Bay group. A number of small stocks have been found intruding both porphyry and sediments.

Granite underlies Great Bear Lake offshore from LaBine Point, outcropping on many islands and at several places on the mainland. The rock is coarse-grained, reddish, and contains a fair proportion of hornblende. It clearly intrudes the Echo Bay group. At normal contacts the granite shows a rough parallel banding for about 10 ft, and numerous partially digested inclusions. Granite tongues have been encountered at a few places in the most westerly underground workings.

Pinkish aplite dikes, varying in width from a few

inches to 20 ft, are frequently seen cutting the granite and older rocks. No aplite has been recognized beyond a thousand feet from the granite.

Intrusives of two or three different ages are identified as diabase. The early diabase includes a number of fine-grained, dark dykes, sometimes amygdaloidal. These are steep in attitude and are known to be earlier than the pitchblende veins. The late diabase is a medium grained, grayish rock of variable texture. It occurs as a flat-lying sill or dyke, showing columnar jointing. The outcrop of the diabase throughout the district is marked by prominent cliffs. Remnants of the diabase in the vicinity of the mine suggest that the dyke formed a continuous domed horizon across the country only a short distance above the present surface. Narrow offshoots of this dyke are found in the mine workings cutting the veins. This is the only intrusive known to be younger than the pitchblende mineralization.

As exposed at the mine, the massive tuff may be 500 ft in normal thickness, while the sediments-porphyry complex approximates 1500 ft. Measurement of a section across the volcanics has never been completed, but the thickness of these rocks undoubtedly exceeds 2000 ft.

Structure: The structure in the Echo Bay group is monoclinal. The strike of the series is northeasterly and the dip is 40° to the southeast, away from the granite. Steep dips and local folding, as at LaBine Point, are found near the granite contact, but average dips decrease leaving the granite until beds are nearly horizontal.

In the mine area the recognized structure involves only the three lowest members of the Echo Bay group, namely, the massive tuff, sediments, and porphyry. These three members are folded on axes striking north-northeast with folds plunging slightly to the north. The details have been determined principally by tracing the porphyry bands, although some confirming evidence has been found by mapping in the sediments. A geological section through the mine would show on the west a syncline with porphyry enclosing sediments to a depth of 1000 ft, and on the east an open anticline with the gently rolling porphyry sheet outcropping over a wide zone.

From the disposition of the folds in respect to the granite, it would appear that the folding can be coupled to the intrusion of the batholith. No such obvious association can be claimed for the major faults in the area, to which the vein fractures are related. These faults fall roughly into two groups, striking east of north and northeast. They have a strong topographic expression and dislocations measured in miles. Numerous lesser faults and fractures are developed in complex pattern over a widely extended area. These "breaks," deeply eroded, form the many narrow valleys, clefts, and crevices which are so noticeable a feature of the topography. The majority have a northeasterly bearing, but some strike south of east, particularly where the fractures fan out in horsetail arrangement from a main break. The faults are often sealed by mineralization of the quartz-hematite type, in extreme cases forming veins hundreds of feet in width.

The period of faulting has been sufficiently extended to affect all the rocks in this area, though the later diabase has suffered only minor displacement. Where diabase cuts through a fault zone there are always certain mud seams which displace it a few feet.

The complexity of the faulting and the brecciated "open" appearance of the fault zones suggest that the movements have taken place at shallow depths. It seems unlikely that the faulting can be ascribed to the deep-seated forces favoring the emplacement of the granite.

The vein-bearing structures are regularly disposed across LaBine Point at about 600 ft intervals, dipping to the north and with strikes converging to the northeast. No. 1, the most southerly, is a strong shear zone up to 40 ft in width, which has been followed by workings and diamond drill holes for over 5000 ft. The apparent horizontal displacement is 300 ft and the movement is left-handed. No. 2 and 3 are structurally minor breaks. They consist of narrow subparallel shearings and related tension members. Displacements do not exceed 20 ft and movements in opposing directions have been observed in different parts of the zones. The northerly vein, called the Dumpy, or No. 5, vein, is another strong shear roughly parallel to No. 1. There are other lesser fractures but all fit into the pattern, fanning out through the block of ground between the main shear zones. This pattern of shearings and supplementary fractures is frequently repeated in the area, but no other group within a radius of three miles has been recognized as of equal economic importance.

Mineralogy: The mineralogy of the pitchblende deposits has been treated in detail by Kidd and Haycock,[4] who have identified over forty metallic minerals and described the paragenesis. The following description is concerned only with observations in the field.

The veins are composed primarily of quartz, carbonate, and hematite in varying proportions with a minor amount of chlorite. Pitchblende, chalcopyrite, nickel-cobalt minerals, bismuth, native silver, argentite, galena, and pyrite are the metallic minerals readily identified. Botryoidal pitchblende was common in the upper parts of the mine but is now rarely encountered. Native silver also was found chiefly in the upper workings. Deposition probably occurred in four distinct stages, and has produced different types of veins depending on the conditions of the fracture zones. In large shear zones stockworks have formed; in narrow fractures there are clean-cut, persistent veins. All veins carry numerous inclusions and, as an extreme instance, No. 3 vein is a breccia cemented by a little vein material.

The No. 1 zone shows the most complex relations. The earliest filling has been quartz with seams and inclusions of chlorite, and a little carbonate. These veins carry some pyrite, but no other metallic minerals. The quartz shows no banding and may, in fact, be hypothermal.

The succeeding quartz-hematite mineralization is the most abundant in the zone. Veins of this type are up to 10 ft in width and form stockworks up to

40 ft wide. The quartz is banded and stained with hematite, forming parallel and concentric structures. Numerous vugs, occasionally a few feet in diameter, are lined with crystals of carbonate, hematite, quartz and chalcopyrite. Carbonate, apparently, constitutes the end phase. Parts of the vein are a dense, solid, rose-red chert. Veins of the hematitic quartz angle across masses of the chloritic quartz. In detail the contacts show intimate penetration into the earlier quartz and it is probable that in the process of refilling the zone the chloritic quartz has been largely replaced.

Narrow bands of sulphides, arsenides, bismuth, silver, and pitchblende are found in the quartz-hematite veins. They show a consistent preference for the cherty phases of the vein, and they are less apt to appear in the prominently banded and vuggy portions. Since these shoots of ore appear in only a small proportion of the quartz-hematite zone, they may represent a separate, third stage of mineralization. On the other hand, the association with certain phases of the hematitic quartz suggests a fairly close relation.

The segregation of the various metallic minerals separately in lenses in the vein is characteristic, although megascopic intergrowths of these minerals on the scale of a hand specimen are not unusual. Thus pitchblende and nickel-cobalt minerals may occur as separate stringers several inches apart, and veinlets of bismuth and calcite may be similarly isolated. This veining with various minerals would appear to offer many opportunities for determining the sequence of the metallic minerals. Such opportunities are rare, however, and the details in observed instances are conflicting.

The metallic minerals are not coextensive. The nickel-cobalt minerals have a wider distribution in the vein than the pitchblende and the latter may occur with only negligible amounts of the other minerals. Silver, though shown by assay to be present throughout the vein in visible amount, is very erratically distributed. It occurs as dendrites in buff carbonate veins, as intergrowths with the nickel-cobalt minerals, and as plates and wires in the pitchblende. On the field evidence some of the silver is surely placed as the youngest of the metallic minerals.

The pitchblende may form solid bands up to a foot and masses up to several feet in width. It is most commonly a persistent, lensing vein a few inches wide, or a lacing network of stringers with some coarse dissemination. The limits of the ore are generally well-defined, and once beyond those limits only an occasional spot of pitchblende is found. Measurements of the radioactivity in the remainder of the vein show that the uranium minerals are absent.

In the last recognized stage in the No. 1 zone, small veins of quartz, carbonate, and chalcopyrite have been introduced, cutting all other veins. Vugs in the older veins have also been filled, or lined with brilliant clusters of crystals. This mineralization is younger than the later diabase. Where the dyke cuts No. 1 zone, it is itself cut by a few veinlets of quartz and carbonate. Carbonate veins in the diabase on Cobalt Island contain metallic bismuth and nickel-cobalt minerals. No silver has been found in the association, but at Gunbarrel Inlet, 40 miles south, a calcite vein cutting the younger diabase does carry silver. It is possible that all the silver may be attributed to this last stage. Galena, which is present with the silver in small amount, may also be related. Narrow veins of galena, normal to No. 1 zone and offsetting its contacts slightly, are clearly a late phase of the mineralization.

Extensive and prolonged movement has taken place on the No. 1 zone. The footwall fault is strongly developed, containing up to 3 ft of gouge. Branch faults lace their way through the zone from side to side creating extremely weak ground. Faulting has continued after the intrusion of the later diabase, as indicated by minor displacements of the dyke contacts. A few veinlets of quartz and carbonate, possibly belonging to the youngest stage of mineralization, are found in the gouge. In a few places, lenses of massive chalcopyrite several feet in width, with a dense and "muddy" appearance, appear to have replaced the gouge.

The Dumpy zone resembles the No. 1 although deposition has taken place on a much reduced scale. The zone is marked by an abundance of chalcopyrite, in irregular masses, of the same type noted in the No. 1 zone.

No. 2 and 3 veins each have their own characteristics. There is no evidence here of the extended period of mineralization recognized in the large shear zones. Various sections of these veins show distinct and different parts of the general mineralization. The openings must have been accessible for relatively brief and critical periods.

In part, No. 2 vein is a beautifully banded carbonate including some rhodochrosite. Pitchblende occurs as seams and botryoidal crusts, and a few sections are rich in silver. Hematite, chalcopyrite, and nickel-cobalt minerals are common, and fluorite may be present. There is no faulting and everything suggests quiet and uninterrupted deposition in an opening probably produced by tension. The remainder of the No. 2 lies along narrow chloritic shearings. Strong quartz-hematite veins carry some of the pitchblende. Other shoots are sparsely mineralized zones in which pitchblende is spotted through the schist accompanied by a few stringers of quartz and carbonate.

No. 3 zone is also sparse in vein material. The breccia, up to 5 ft wide, is sharply defined and contains numerous fine and coarse angular fragments in a dark matrix. Disseminated pitchblende and chalcopyrite, occasional masses of pitchblende, and a very few quartz and carbonate veinlets have been introduced. The breccia zone may die out leaving only a thin gouge seam to mark the zone, but the pitchblende may persist as spots in the gouge. The pitchblende in No. 3 forms a high proportion of the total mineralization and this is true of many other minor occurrences in the district.

The characteristics of these veins are those of deposits formed under moderate conditions of pressure and temperature. The veins seem to belong in the upper part of the mesothermal group.

An unusual feature of the mineralization are the coarsely crystalline aggregates which replace sedi-

ments, and porphyry to a minor degree, in highly irregular fashion. These aggregates consist of amphibole, epidote, apatite, carbonate, and magnetite, with lesser amounts of garnet, feldspar, chlorite, and pyrite. The skarn mineralization is not obviously related to any of the veins nor to the altered zones, but the amount of radioactivity within the areas of occurrence implies some connection; perhaps it is a product of reaction between lime-rich phases of the sediments and the invading solutions.

Alteration: The alteration in the mine area is prominent, and the "baked" appearance of all formations, except the later diabase, has been frequently mentioned. Both the alteration on the veins and the more general alteration are of the same type, and result in wide-spread discoloration of the rocks by hematite and the obliteration of original textures. This so-called red alteration, undoubtedly related to the quartz-hematite period of mineralization, affects the quartzose rocks most severely, but where alteration was intense there is little selectivity. The rocks are then reduced to a dense reddish "jasperoid." The exact nature of the alteration has not been determined, but quartz, hematite, magnetite, sericite, chlorite, and carbonate are obvious constituents.

Away from the mine the degree of alteration falls off, but the mineralization is so pervasive that in no case can it be said that examination has been carried out of the zone of alteration. The distribution of the alteration points to the mine as being a center of mineralization in the district, and indicates that the veins and alteration have a common hydrothermal source. In a geophysical examination of the area, A. A. Brant has noted an unusually high magnetite content of the rocks on LaBine Point. The percentage of magnetite decreases away from the mine, thus giving further evidence as to the locus of mineralization.

The envelope of reddish "jasperoid" enclosing the veins is the most intense form of alteration. Within 4 or 5 ft of the vein there are seldom even recognizable remnants of the wall rock. Although the width of the altered zone is roughly proportional to the width of the vein, there are examples of unusual penetration, probably on numerous small mineralized fractures. The red alteration may mark areas in which veins are concealed and is therefore a useful guide in exploration.

In the larger shearings, in addition to the red alteration, there are strongly developed talcose and chloritic zones. The chloritic phase is poorly veined and has probably not been long accessible to the altering solutions. The buff and reddish talcose phases are various advanced stages accompanying abundant vein deposition.

Factors Controlling Ore Deposition: The ore bodies are, for the most part, narrow lensing streaks of pitchblende in a much larger volume of vein material. Shoots of minimum stoping width range in length from 50 to 700 ft, and have been followed vertically for more than 600 ft. The ore bodies locally widen to as much as 15 ft because of the occurrence of multiple stringer zones and masses of pitchblende.

Two of the important factors in ore deposition are the wall rock, and the nature of the openings. The best evidence of their influence at Eldorado is offered in the case of veins in narrow fractures. In the large shear zones, where the proportion of total vein material to ore is very high, it is difficult to demonstrate any effective control.

The optimum of control must have existed during the period of formation of No. 2 vein. The zone is a composite of shear and tension members, over 2000 ft in length, each part of which carries some ore. The rocks intersected by the zone are sediments, porphyry, and early diabase. Most of the ore shoots have been found in sediments apparently favoring the finely banded phases. Some ore occurs where the shear has followed the contact of the sediments and early diabase. No ore is found in the porphyry except as the ends of the larger ore shoots in the adjacent sediments. In longitudinal section the termination of the ore shoots at the trace of the porphyry contact on the vein is very well shown. Thus, while different types of sedimentary rocks as well as the early diabase are favorable to ore deposition, the porphyry is decidedly unfavorable.

The noticeable constriction of the fractures on entering the porphyry has not resulted in the exclusion of vein material and does not, therefore, fully explain the absence of ore. The sediments have been affected by vein alteration to a much greater degree than the porphyry. They must have exercised a correspondingly greater influence chemically on the vein-forming reactions. The more favorable chemical environment of the openings in the sediments and the restricted passage of solutions through the porphyry have probably combined to localize the ore shoots.

The greater proportion of ore in No. 2 occurs in the tension member. This fracture is filled with a well-banded carbonate, carrying better than average percentages of pitchblende. The principal carbonate mineralization is a late phase of the quartz-hematite stage, and the tension fracture must have opened up during the invasion of the carbonate solutions. Not only did the fracturing occur at precisely the right time, but also in view of the grade of the vein, conditions must have been ideal for pitchblende deposition. There are many similar pitchblende showings in the area, too small to be minable. In these showings 1 to 3 in. of vein material may fill a well-defined tension fracture for 20 ft or so. Pitchblende may amount to as much as 50 pct of the vein and in every case the proportion of pitchblende to vein material is higher than average.

The tension fracture in No. 2 zone has an interesting relation to the shear members and also to the local rock structure. The zone cuts across a small syncline of sediments lying in porphyry. The tension fracture is at a low angle, both on plan and section, between two parallel shearings. At the same time it is confined to the sediments, extending from the porphyry on one flank to the porphyry on the other. Maintaining the same relative positions on successive levels, the fracture shortens appreciably with depth and will presumably die out when the bottom of the sedimentary trough is reached.

In No. 1 zone, the large veins have been localized by slight changes in attitude of the shear. At least this is suggested by development work to date on

No. 1 zone. It will require much more development, fixing the position of the vein accurately on several levels, before definite conclusions can be drawn. The ore shoots within the vein occur without regard for the wall rock. It is possible that ore deposition has been related to certain phases of the quartz-hematite vein but no systematic arrangement of the ore shoots has resulted. Most of the pitchblende is found on or near the footwall, but other lenses are so disposed across the zone as to require thorough exploration from wall to wall.

Source of the Ore Solutions: It is usually accepted that ore deposits are related in time to one or another of the intrusives in an area. Locally a closer relationship is suggested by the distribution of the ore in relation to an intrusive. No evidence was found at the mine to link the pitchblende deposits closely to one particular intrusive. The more general consideration, that the ore solutions have had a common origin with the granite, is open to several objections.

In the first place, the faults and fractures have been formed at shallow depths as shown by their brecciated and open appearance, a fact which contrasts with the deep-seated emplacement of the granite. Second, the faulting is regional in its distribution and cannot be related solely to the mass of granite lying west of the mine. Third, the faults displace several intrusives younger than the granite thus introducing the likelihood of a considerable interval between the granitic intrusion and faulting. Four, some of the faulting cuts the later diabase and, if these final movements are attributable to the same forces, the interval is great enough to make improbable any genetic relationship to the granite.

The major faults have tapped an abundant source of solutions, to which the "giant" quartz veins bear evidence. If granite differentiates have originated at the same time they have not been recognized.

The later diabase can be coupled to the ore deposits more closely than any other intrusive. Some of the silver is known to be later than the diabase and the association is elsewhere accepted as evidence of a common source. The diabase, however, clearly intersects the pitchblende veins. It has not been subjected to the general alteration. There are no general considerations supporting the proposed relation between pitchblende ores and diabase.

The conclusion is that during an extended period of faulting solutions from an undisclosed source have risen to form the relatively shallow-seated pitchblende deposits. This period of mineralization came to a close but activity was renewed during intrusion of the later diabase with the consequent deposition of silver.

International Uranium Ltd.—by W. F. James and A. H. Lang

Nine miles south of the Eldorado mine a native silver-pitchblende deposit was operated intermittently prior to the war. The deposit consists of three veins containing mineralization of the same general character as that of the Eldorado mine, but occurring in granodiorite near the contact of a younger granitic intrusion.

A 25-ton concentrator is reported[15] to have produced 6933 lb of U_3O_8 in 1938 and 1939. The present company, called International Uranium Mining Co. Ltd., was incorporated in 1942. Operations were suspended for a time, and the mill was destroyed by fire. In 1947 the company shipped high-grade silver ore. Some pitchblende ore shoots were outlined in old workings. Shaft sinking was resumed in 1948, with a view to driving two deeper levels in the hope of outlining additional pitchblende ore.

Goldfields Area, Saskatchewan by S. Norman Kesten

The presence of radioactive minerals in the Goldfields area was recognized first in 1935 on the present Nicholson property, two miles east of the former mining town of Goldfields on the north shore of Lake Athabaska (fig. 1 and 2). A deposit of pitchblende was found while the property was being explored for copper, and although some work, including the driving of two small adits, was done on the showing, it aroused little interest at that time. However, during World War II, when intensive exploration for uranium minerals was undertaken, the Nicholson again received attention. In September 1944, the showings were examined by A .W. Jolliffe and R. F. Murphy. The former represented the Geological Survey of Canada while the latter acted for Eldorado Mining and Refining (1944) Limited, the Crown company formed to mine the famous pitchblende deposit at Port Radium on Great Bear Lake. As a result of this examination, Eldorado staked forty-seven claims adjoining the Nicholson property on the north and east.

Detailed exploration of these claims was carried out during the field seasons of 1945, 1946, and 1947, including 24,000 ft of diamond drilling. At the same time, the camp established at Fish Hook Bay as a base for this work also served as a base camp for expanding operations within a 20 mile radius of Goldfields. By the end of 1948, the Crown company had staked 193 claims in the area and had carried out detailed exploration on about two-thirds of the ground covered. Except for recent activity on the part of the Nicholson company, Eldorado has done virtually the only work on radioactive deposits in the area, and the following remarks will deal almost entirely with the latter company's operations and findings.

General Geology: *Formations:* The rocks of the Goldfields area are of pre-Cambrian age. The oldest are those of the Tazin group, consisting of dolomite, quartzite, argillite, conglomerate, flows, and fragmental rocks. There is some doubt as to whether these are Archaean or Proterozoic. Their deposition was followed by intrusions of granitic rocks. On these were laid the sediments classed by some writers as the Beaverlodge series, which in turn were intruded by basic rocks. A second granite intrusion was followed, apparently after a long period of erosion, by deposition of the sediments of the Athabaska series, which is Proterozoic. This sedimentation was accompanied by the extrusion of basic lavas and the intrusion of small basic dykes. These are the youngest rocks in the area; erosion has brought the land surface down approximately

to that which existed at the time of their deposition so that there remain only remnants of the Athabaska where these rocks were deposited in basins or subsided in the course of late faulting and folding.

Structure: Earth movements appear to have followed each period of sedimentation outlined above. Structurally the area may be divided into four parts, each a separate geographic district (fig. 2).

1. The Cornwall Bay district lies along the north shore of Lake Athabaska, from Elliott Bay on the west to MacIntosh Bay on the east and includes Goldfields, Cornwall Bay, and Fish Hook Bay. Here Tazin and Beaverlodge sediments have been folded in a broad, open syncline, plunging gently to the south, forming a semicircle about the town of Goldfields. The folded formations are mainly thick beds of quartzite and dolomite which have been intruded by gabbro sills and by the younger, pre-Athabaska granite.

2. The Beaverlodge Lake district embraces Beaverlodge and Martin lakes which are to the north and west of Goldfields. The margins of Beaverlodge Lake are underlain by conglomerates and sandstone of the Athabaska series which, in most cases, appear to have been deposited in erosion troughs in the older sediments. The basin in which Martin Lake lies is formed by a syncline of Athabaska sediments interbedded with basalt flows, plunging gently a few degrees east of north.

3. The Eagle-Ace lakes district is north of Beaverlodge Lake and south of Mickey and Fredette lakes. It is underlain by Tazin rocks which include graywackes, basalts, and possibly some tuffs and basic sills. There is some of the older granite in the northern part of the district and most of the quartzite has been highly granitized. The rocks, in addition, have been severely crushed, folded, and squeezed on several occasions, so that they now appear as a series of tight, overturned folds striking northeasterly. The formations are cut by members of several sets of faults or "zones of movement" which also strike northeasterly.

4. The Donaldson Lake district is south of the lake of that name and between Ace Lake on the west and Raggs Lake on the east. The situation here is similar to that in the Eagle-Ace district except that there is more granite, granitization is more intense, and arkoses have been so altered as to be almost indistinguishable from granites and granitized sediments. Faults have been masked by later alteration, but that there was much movement is evidenced by the widespread development of amylonites. One fault, later than most of the alteration, is of great importance in the district. This is the St. Louis fault, which strikes N 65° E from Beaverlodge Lake, through Ace Lake to Raggs Lake, gradually swinging to the north, having a strike of about N 55° E at Christie Lake. It is a gravity fault dipping 45° to 65° to the south. Between Beaverlodge and Ace lakes down-faulted Athabaska sediments have been protected from erosion by the movement.

Economic Geology: Deposits of pitchblende of economic interest are both numerous and widely scattered in the Goldfields area. The Eldorado company has found well over a thousand radioactive anomalies on its claims within ten miles of Goldfields. The deposits studied in detail have many characteristics in common.

Pitchblende has been found in fractures in virtually all the rock types in the area and is younger than any known intrusion. While not confined to any type of rock, the veins are richer and better developed in basic rocks, including limestone and dolomite. The veins were deposited in open fissures at shallow to moderate depths. Characteristics of the fractures indicate that, for the most part, they owe their origin to tension. Carbonate, mainly calcite, is the common gangue mineral, and it usually has associated with it small quantities of hematite. With a few notable exceptions the only other mineral associates are very minor amounts of quartz and chalcopyrite. Except in some of the deposits along the shore of Lake Athabaska, the wall rocks of veins have been altered characteristically to a red cherty material, the result of the addition of feldspar and hematite.

Cornwall Bay District: Occurrences on Eldorado's Fish Hook Bay and Gil properties and on the Nicholson ground have been intensively explored. Pitchblende has been found on the Box gold property of the Consolidated Mining and Smelting Co., and anomalies are said to have been found by prospectors during 1948 on ground immediately north of Cornwall Bay.

1. On the Fish Hook Bay property almost 400 separate radioactive anomalies have been found, and pitchblende is visible in about 15 pct of these. The pitchblende-bearing veins exposed on surface are confined to dolomite, dolomitic iron formation and dolomitic quartzite of the Tazin series. They are in tension fractures striking subparallel to the topographically prominent Fish Hook valley. In one vein masses of botryoidal pitchblende are accompanied by massive arsenides of cobalt and nickel. At two other occurrences, masses of an unidentified vanadium mineral were found. None of the veins exposed on surface proved to be of sufficient extent to warrant further investigation at present. However, the larger portion of the 24,000 ft of diamond drilling carried out on the property was concentrated in Fish Hook valley. The bay and the muskeg were found to cover beds of quartzite and siliceous iron formation which have been severely crushed and then cemented by red hematite and calcite. The brecciated hanging-wall and footwall rocks of this westerly-dipping, northerly-striking crushed zone are locally well mineralized with pitchblende. It has been impossible at the present stage of development, to define clearly one or more distinct veins, or zones of mineralization, but further work may prove the pitchblende to be present in minable concentrations.

2. On the Nicholson property work has been done on only a few of the radioactive anomalies found. The geology of the individual occurrences is not well known to the writer, but most of the deposits exposed appear to be related to bodies of crushed and hematitic quartzite similar to but smaller than that at Fish Hook Bay, one half mile to the east. The deposit presently considered to be the most im-

portant lies in or near the contact between a crushed zone on the east and Tazin sediments on the west. The crushed zone underlies a valley parallel to the Fish Hook valley. When last seen by the writer the vein had been explored by trenching for a length of 500 ft and its presence detected by Geiger-Müller counter for a length of 1000 ft. It was later stated to have been trenched at intervals for a total length of about 1000 ft. It is pierced by two adits, vertically 60 ft and laterally 200 ft apart. Mineralogically the vein is unique among those in the Goldfields area. The following remarks are quoted from a report dated October 6, 1944, written by R. Murphy.

The pitchblende occurrence is similar to that at No. 1 Vein at Eldorado, with a gangue of red-banded quartz, carbonate and platy hematite and the usual related nickel-cobalt minerals concentrated with the pitchblende in narrow seams in the ore. Textures in the ore are similar, with botryoidal and crustified structures exhibited by the sulfarsenides and gangue. The single divergent feature is the presence of thucholite at the Nicholson.

It is worthy of note that samples from this vein gave significant values in gold and platinum.

3. At the Box property fragments of pitchblende have been found in the overburden within about 2000 ft of the main shaft of the mine, but the occurrences have not been investigated as far as the writer knows. In addition, in 1946, members of a party of the Geological Survey of Canada found many anomalies in the outcrop of the ore body. These were associated with narrow fractures, a few feet long, in the granite. The walls of the fractures are stained a deep red. No mineralization was found.

4. On Eldorado's Gil property there are nine pitchblende-bearing veins in flat-lying gabbro sills intruding massive quartzite beds of the Beaverlodge series. The veins are deeply weathered and fresh material has not been exposed, but the veins are believed to conform to the general pattern outlined at the beginning of this section. While the veins are confined to the sills, surface evidence suggests that the fractures in which they were deposited persist from one sill to another, through the intervening qu :tzite. Surface evidence also suggests that there are two or more sills sufficiently thick to give the veins considerable promise. However, diamond drilling results were disappointing: the uppermost sill appears to have a maximum thickness of 50 ft and the lower sills are much thinner.

Beaverlodge Lake District: Occurrences of radioactive anomalies on five of Eldorado's properties near the shores of Martin and Beaverlodge lakes have been explored by surface operations, and one of them by underground workings. The pitchblende-bearing veins on these properties all lie in fractures in rocks of the Athabaska series. They conform closely with the general description of veins in the area. The fractures trend easterly to northeasterly. The properties are the Beaver, Westy, Murphy, Ra, and Ura. Of these the last two deserve separate discussion. On the other three the veins rarely exceed 50 ft in length, show values in pitchblende insufficient to render them individually of economic importance, and are not sufficiently closely grouped to enhance their possible value.

1. On the Martin Lake property (Ra), at the east shore of Martin Lake, a series of veins in a fracture zone several hundred feet long is being investigated from an adit. Exploration of the zone, first found on surface by methods to be described, has shown the possible existence of minable concentrations of pitchblende. The fracture zone is parallel in strike to the St. Louis fault, mentioned in the section on structure, and although it is not asserted that the fault may be traced this far west, it seems likely that the fracturing is related to it. Here, as elsewhere, the mineralization favors the basic rocks and is found almost exclusively in the basalt flows, only rarely in the sandstone beds.

2. On the Ura property, 2½ miles to the east of the Ra, on the shore of Beaverlodge Lake, a series of parallel fractures, some of which are 200 ft long, occur in a zone more than 2000 ft long. Diamond drilling 1400 ft in this zone gave inconclusive results, but it appears that the mineralization found on surface does not persist to a depth of 100 ft. It may not continue from the shallow Athabaska conglomerate, in which it occurs on surface, into the underlying Beaverlodge quartzite. Both the zone of fracturing and the individual fractures within it are parallel to the St. Louis fault. The occurrences are about 1000 ft south of the fault.

Eagle Lake District: The block of ground explored in this district is west of a north-south line through Eagle Lake, extending more than a mile both north and south of the lake and about a mile and a half to the west. Approximately 200 anomalies, a large proportion with visible pitchblende, have been found here. Many of the veins have been exposed by trenching and one large group of occurrences has been explored by diamond drilling.

1. The Ato showings are in the southern part of the district. They consist of several pitchblende-bearing veins in fairly flat-lying slaty sediments. They are believed to be in tension fractures related to inferred faults striking northeasterly. The fractures strike at a large angle to the strike of the faults. If the mineralization is confined, or almost confined, to basic rocks, as experience elsewhere suggests, its importance may be greatly restricted by the thickness of the slaty sediments.

2. The Eagle property in the center of the district has been explored by about 22,000 ft of diamond drilling. Here again the veins are found for the most part in the basic lavas and sediments and, where values are found in the granitized quartzite, they are usually within 10 ft of a contact with less acid rock. The formations here are more favorable to mine-finding than they are a mile to the south; the beds of basic rock are up to 100 ft thick and dip fairly steeply. The diamond drilling results suggest, however, that there are many short veins and only a few up to 300 ft in length.

3. The Mic showings are at the north end of the district. Deposits of pitchblende which appear important were found at the base of a scarp which marks a northeasterly-trending fault. The mineral is in a swarm of small irregular fractures within 40 ft of the fault. The strike of the fracture zones is about normal to the strike of the fault. The show-

ings have been stripped only, therefore no unaltered mineralization has been seen, but there does not appear to be much gangue accompanying the pitchblende.

Ace-Donaldson Lakes District: This subarea lies between Beaverlodge Lake on the west and Raggs Lake on the east. It is limited on the north by Donaldson Lake and extends a few thousand feet south of the St. Louis fault. Most of the occurrences lie either along the fault or within 2000 ft to the north or south of it. The fractures in which the veins are found are usually parallel or subparallel in strike to the fault. The deposits on the Ura property, discussed above, might be included in this group. Three other properties are discussed below.

1. On the Ace property the St. Louis fault dips 48° to the south and separates basic rocks of the Tazin series in the footwall from conglomerate and minor sandstone beds of the Athabaska series on the hanging wall. On surface some anomalies were found in the conglomerate but most promise was given by showings in the footwall rock. A number of short and narrow carbonate veins, rich in pitchblende, occur in tension fractures subparallel to the strike of the fault and within 100 ft of its surface trace. One important vein appears to dip at 38° towards the fault but the majority of the veins dip steeply in the same direction. The fault has now been pierced by 28 diamond drill holes, drilled from the hanging wall into the footwall. The core has yielded some high values in the U_3O_8, extending to a hole depth of about 100 ft below the fault. In some instances the high values are confined to streaks and stringers of pitchblende, whereas in others the radioactive mineral appears to be disseminated thinly throughout a body of intensely-altered, red rock. The pitchblende is in very small and irregular fractures of the tension type and is accompanied by red calcite.

2. The Bolger property is of interest for a reason additional to the probable value of its veins. Over an area of 6500 sq ft, the overburden, which is a few feet deep in most places, is rich in uranium in the form of the bright yellow and orange hydrous oxide. This rich overburden lies in a slight depression in the slope of a hill. It is suggested that rainwater collected in the shallow basin, leached uranium from underlying veins and deposited it throughout the overburden in the form of the oxide. Several trenches were dug at the locality but consolidated rock was reached at only a few places. The presence of pitchblende-bearing carbonate veins in fractures parallel to the strike of the St. Louis fault was indicated by this work.

3. The Donaldson Lake zone is about 10,000 ft long and up to 400 ft wide. The zone is subparallel to the strike of the St. Louis fault and about 1000 ft north of it. It extends southwesterly from a point 2000 ft east of Donaldson Lake. Here anomalous radioactivity is often found to emanate from the surfaces of large outcrops of rock, but the activity is usually localized in fractures trending parallel to the strike of the fault. The fractures occasionally hold coarse white carbonate and minor quantities of pitchblende. The original character of the host rock is masked by replacement by coarse red feldspar in which there are sheared shreds of green chlorite. The interstices between grains of feldspar are filled by white carbonate. The walls of the fractures are usually a deeper red than the remainder of the rock. It is considered that the rock in a large shear zone has been replaced by the feldspar and minor carbonate and that these minerals were accompanied by pitchblende. The economic possibilities in this zone have yet to be explored, but there is some evidence that it contains one or more of the vein type of deposit as well as large, low-grade bodies.

Other Occurrences: In addition to the occurrences discussed above, many widely scattered anomalies have been found within 20 miles of Goldfields. Few of these have been thoroughly explored. The majority of them are in or near bands of basic rock and are associated with pronounced topographical features which trend northeasterly. The veins, some of which are pitchblende-bearing, have characteristics similar to those already described.

Methods of Exploration: *Introductory: the Geiger-Müller Counter:* Prospecting for and the exploration of radioactive mineral deposits differs from the same operations in connection with, say, gold deposits, chiefly in that work with the radioactive minerals is greatly facilitated by the use of the Geiger-Müller counter. The instruments used by the Eldorado company were developed by the National Research Council of Canada and adapted for several different purposes. The use of the counter has served to enlarge the visual sense of those employed in exploration. It will detect the presence of radioactive minerals where they cannot be seen by the unaided eye, even when the mineral is beneath a few feet of overburden or several inches of rock.

Prospecting: In the Goldfields area prospecting was carried out by experienced men who employed normal methods of prospecting but who were greatly aided by the use of counters. The model developed for the purpose is light, rugged, simple to operate, and requires little servicing. The tube is with the other parts of the instrument in a small box weighing about 7 lb. There are but two adjustments for the operator to make: he turns it on and switches it off, and increases the voltage as the batteries weaken. The instrument has no other signal but that heard through earphones. The batteries which provide the power have a life of six weeks to two months and are simple to change when worn out. The instrument is slung from a shoulder strap so that the tube is at waist height. For closer inspections the instrument is held a few inches above the ground. It is probable that few, if any, of the prospectors' many discoveries in the area would have been made without aid of the counter.

Geophysical Surveying: After a discovery has been made and claims have been staked, the next step in exploration is a detailed radioactivity survey of the staked ground. The survey consists of close examination of the ground with a portable counter expecially adapted for the purpose. It is similar to the prospectors' model, but a little larger and 2 lb heavier. It is slightly more sensitive, having a 5 in. Geiger-Müller tube instead of a 4 in. one. A late model contains three synchronized 6 in.

tubes. Both models have microammeters by which roughly comparative readings may be taken.

The survey of a property entails traversing the ground at intervals of 15 ft. Although specimens of pitchblende 4 or 5 lb in weight lying on the ground may be detected by a counter operator at a distance of 10 ft when the instrument is carried at waist height, a vein under a few inches of soil may be undetected at a distance of 7 or 8 ft. Consequently re-surveys generally disclose additional deposits, as many as 50 pct of the number found in the original survey, even when both surveys are made with traverses at 15 ft intervals.

When the operator detects an anomaly by an increase of the rate of clicking in his earphones and confirms it by noting an appreciable increase on his milliammeter scale, he holds the instrument close to the ground surface and attempts to trace out the dimensions of the anomaly. Then he uses a light pick to uncover the source of the radioactivity, if possible. Finally, he locates the discovery relative to chainage points on picket lines cut on the property previous to the survey. The discovery is reported to a geologist who examines it to determine its possible importance. Exploration from this point proceeds according to usual practices.

Uranium minerals deposited in overburden occasionally give misleading results in surveys of this kind. Furthermore, it is very desirable to make such surveys before any blasting is done, as fragments of pitchblende scattered by blasting will "salt" a large area.

Diamond Drilling: Special Aids: The drilling of groups of deposits has taken place following stripping, trenching, and sampling in some instances. In other instances, however, drilling has followed immediately after the geophysical survey and geological examination, because it has not been anticipated that trenching of the deeply weathered carbonate veins would succeed in exposing fresh material at economic depths. Instead, the evidence given by the counter as to the value of the occurrences, together with the geological relationships, has been taken as sufficiently reliable.

Interpretation of drilling results has been greatly aided by adaptations of the Geiger-Müller counter for logging the core and for logging the holes.

The drill core is logged by two different forms of the counter. In the first method a sensitive, portable, battery-operated counter, normally used in the radioactivity surveys, is shielded by lead blocks 2 to 3 in. thick and arranged in such a way that a box of core may be passed close to the unshielded Geiger-Müller tubes. This method results in the detection of radioactivity in the core equivalent to that emanating from a concentration of as little as 0.05 pct U_3O_8, but is usually reliable no lower than 0.10 pct U_3O_8. Recently, more satisfactory results have been obtained from a line-operated instrument with a 6 in. tube in a probe at the end of a short cable. The probe is placed among the lead blocks much as the portable instrument was, and from there scans the core in the same manner. The line counter consistently detects concentrations of less than 0.03 pct U_3O_8.

The instrument employed to log drill holes is powered by batteries. The tube is in a probe at the end of a cable several hundred feet long. The probe is dropped, or pushed, down the drill hole, and ammeter readings are taken at 5 ft intervals. The instrument is equipped with a loud speaker which instantly signals an increase in radioactivity. When this happens readings are taken at 1 ft or ½ ft intervals. After three years of operation sufficient data have been gathered to make it possible to relate scale readings to values obtained from the core. Where high readings have been obtained in the drill hole and no anomalous radioactivity has been found in the core it is possible to interpret the readings in terms of the probable concentrations of U_3O_8 in the walls of the drill hole. The diameter of the drill core is thus enlarged, in effect, from 1⅛ in. to approximately 1 ft.

Tobey-Albrecht Deposit—by W. F. James and A. H. Lang

The Tobey-Albrecht property at Black Lake (fig. 1), 110 miles east of Goldfields, was discovered in 1948 by two prospectors working under the Saskatchewan government assistance plan. A concession 2½ by 12½ miles in size was granted to the discoverers, who sold it to Transcontinental Resources Limited, a company that also controls the Nicholson and Camray properties. Other nearby concessions were acquired by various individuals and companies.

The mineralization occurs in quartzite and schist mapped tentatively as part of the Tazin Group, of Archean age. The mineralization occurs in a shear zone and in fractures, all lying almost immediately west of a prominent northeasterly-trending fault, called the Black Lake fault, that has been mapped for several miles.

The occurrence is described as follows by Dr. A. R. Byers,* who examined it for the Saskatchewan government:

* Unpublished report quoted by permission of Saskatchewan Department of Natural Resources.

At the time of the writer's visit, two areas of radioactivity approximately one mile apart along the strike of the fault zone, had been discovered.

The south showing consists of a mineralized shear, 18 to 24 inches wide which follows the contact between a foliated, fine grained amphibolite on the west and aphanitic, cherty sediments cut by aplitic and pegmatic stringers on the east. The shear strikes 045° and dips 85° southeast to vertical. The Geiger counter indicated a radioactive content of two times normal background for a length of 25 feet along the shear. This occurrence lies east of the main fault and cuts the sediments belonging to the Tazin group. It is the only occurrence of radioactive mineralization noted east of the fault and lying in these sediments.

The main or north showing, about 800 feet southwest of a large sandy point, is a zone of fractured gneiss approximately 2400 feet long and 200 to 500 feet wide. This zone lies just to the west of and parallels the strike of the main fault.

The majority of the fractures fall into three sets: (1) a set strikes and dips approximately parallel to the main fault and represents small subsidiary planes of shear. (2) a second group of fractures strikes 065° to 080° and dips 75° to 85° northwest. This also represents a direction of shear as indicated by slickensiding and sheared rock along the fractures. (3) a set of transverse fractures strikes 115° to 125° and dips 75°

northeast to vertical. The fractures of this set have the characteristics of tension structures.

If the strain ellipsoid is orientated on the basis of the regional fault being a direction of major shear, then the composite fracture pattern formed by the above three sets of fractures agrees closely with the theoretical fracture pattern to be expected.

Individual fractures can be traced along the strike for distances of 50 to 200 feet, a few up to 500 feet. The fractures contain little vein material but are characterized by a reddish-brown to red discoloration of the country rock and a partial to complete loss of definition of the crystal texture of the gneiss. This zone of alteration extends out into the rock on either side of the fractures for a distance of a few inches to several feet.

The only metallic minerals noted are pyrite, cobaltite, and pitchblende. Pyrite is sparsely present in the wallrock along the margins of the fractures. Cobaltite was noted at only two places and appears to be very rare. Pitchblende occurs as small lenses and stringers, a few inches to several feet long and one-eighth to three inches wide, distributed at irregular intervals along some of the fractures.

Ten character samples taken by Dr. Byers assayed from 0.032 to 21.42 pct U_3O_8 equivalent.

The Camray Deposit

The Camray discovery is at the shore of Theano Point of Lake Superior, 70 miles north of Sault Ste. Marie (fig. 1). It is about 1½ miles from a highway that extends from Sault Ste. Marie to Montreal River.

The history of the discovery has already been discussed in the historical section of this paper. Because the find was made late in the season, no work has been done yet to expose the showings, but a road has been built to connect them with the highway, and it is expected that exploratory work will begin on this and other discoveries in the neighborhood as soon as the snow melts.

The deposit is described in some detail because it is the only pitchblende occurrence in eastern Canada for which much information is available.

General Geology: The region is in the southern part of the pre-Cambrian Shield and the Camray claims are underlain by pink granite and granite-gneiss probably Algoman in age. The rock is porphyritic in places, and the quartz in some specimens is so fine-grained that the rock resembles syenite to the unaided eye, a fact that probably explains LeConte's mention of syenite. The granite contains dykes and irregular masses of pegmatite.

The granite is intruded by diabase dykes, only one of which was studied. It consists of fine-grained, considerably altered diabase, which is much jointed and which tends to weather in blocks 2 to 6 in. in size. The diabase dykes of the region are classed as Keweenawan in age and are of two ages: an older, olivine-free type, and a younger, olivine-bearing type. Study of two thin-sections of specimens from this dyke failed to show the presence of olivine, therefore the dyke is classed tentatively with the older group.

Mineralization: The mineralization consists of quartz, pitchblende or coracite, carbonate, mica, and secondary uranium minerals. A sample representative of the 6 in. lens mentioned below, indicated a content equivalent to 8.72 pct U_3O_8. Coracite is listed in early editions of Dana's "System of Mineralogy" as a variety of uraninite showing partial alteration to gummite. The seventh edition lists coracite as synonomous with gummite, and states that the name gummite is used in that edition as a generic or field term for substances essentially oxides of uranium. Satterly[10] reports that three specimens from the recent discovery were examined by E. W. Nuffield of the University of Toronto, who obtained X-ray diffraction patterns identical with those obtained from pitchblende from Great Bear Lake. Nuffield and Satterly, therefore, class the Camray mineral as pitchblende. Unless further mineralogical or analytical studies indicate that the name pitchblende is not applicable, it will probably be best to use that term and to discontinue use of the name coracite. The mineral is called pitchblende provisionally in this paper.

Description of the Deposit: The discovery consists of zones of pitchblende-bearing stringers that fill fissures in the granite walls immediately adjoining the diabase dyke described above. Because the erosion of the dyke has left a canyon, the stringers are exposed on almost vertical cliffs. The upper parts of the cliffs, which are about 50 ft high, cannot be examined at present, nor can the extent of the stringers into the walls be ascertained. The dyke and canyon strike N 80° W and have an average width of 35 ft, but at one place the dyke narrows to 18 ft and the remaining width of the canyon is floored by granite. The dyke dips 70° to 80° northward.

Deepening of the canyon at the shore of the lake has caused the formation of a small cove, where the water normally extends inland about 140 ft from the general shoreline of Theano Point. The high-water mark is about 200 ft farther inland. Eastward from the cove the floor of the canyon rises gradually for about 500 ft, where it is perhaps 50 ft above lake-level; beyond this point the floor rises steeply. The dyke outcrops intermittently along the flatter part of the floor, and probably continues much farther inland but is covered by overburden.

The greatest amount of pitchblende is exposed in a lens up to 6 in. wide, which tapers to a thin stringer within a length of about 3 ft. Elsewhere, the stringers range from ⅛ to 2 in. in width, most of them being ½ in. or less. Usually they are only exposed for lengths of 2, 3, and 4 ft. Many of the stringers strike N 70° E to N 85° E, and dips range from 35° SE to vertical. Because the pitchblende is brittle and easily eroded, often the fractures are now unfilled for a depth of ½ in. or more from the face of the cliff; there pitchblende cannot be seen but high counter readings indicate the likelihood of its presence.

The stringers are grouped in two main zones along the footwall or south side of the canyon, the one nearer the shore having an overall length of 275 ft, and the other 100 ft. These are separated by an apparently barren zone 200 ft long. At the east end of the shoreward zone of stringers a small fault appears to displace the dyke about 2 ft. This fault may have some bearing on the narrowing of the dyke nearby, but the relationships are not yet well exposed. No pitchblende was seen in the fault,

but high counter readings indicate that it may be present. This is the only indication that stringers may occur in the dyke.

About 400 ft east of the more easterly zone of stringers three, high counter readings were obtained within a length of 80 ft along a granite cliff that appears to be the continuation of the footwall. No pitchblende was seen, but development work here might disclose another zone of stringers.

On the north or hanging-wall side of the canyon two stringers ⅛ to ¼ in. wide and 4 ft long are exposed at a locality about 130 ft east of the head of the cove. One of these is perpendicular to the dyke and the other is parallel with the contact between the dyke and the granite.

Mr. Campbell stated that a high count is obtainable in the bottom of another gorge, about 1/3 mile east of Theano Point, that may represent the weathering of a dyke oblique to the direction of the one described above. It would be interesting to trace the continuation of the main dyke to its intersection with this gorge, on the chance that pitchblende might be concentrated there. Snow prevented the writer from checking the reading at this locality.

Grade: The deposit is not yet exposed in such a way as to permit sampling of a type that would give any indication of the grade of a ton of the stringer-containing granite. To cut samples along the cliff faces would not be practical for two reasons: because of the weathering of the stringers, and because such samples would be parallel with, rather than across, the probable direction of mining.

The field Geiger counter cannot be used in an absolute, quantitative way for estimating the grade of a deposit such as this, because the needle goes to the limit of the scale when held against the larger stringers, and because, when held against rock a foot or two away from a stringer, the reading represents rays coming from the stringer rather than from the area of contact.

Samples of the stringer material itself indicate the character of the mineralization, but not minable grade, consequently the results of such samples are included in the mineralogical section of this report.

Samples of the granite near the stringers, when tested with a quantitative, laboratory counter, indicated an average radiation equivalent to 0.006 pct U_3O_8, and specimens of the dyke rock gave an average of less than 0.001 pct U_3O_8. As counts of this order are commonly obtained from rocks, it does not seem likely that the rock between the stringers will prove valuable. If the deposit proves to be minable, it will probably be necessary to sort the stringer material from the country rock.

Origin of the Deposit: The pitchblende probably was deposited from hydrothermal solutions of magmatic origin. The relationship to the dyke seems to be entirely structural. The mineralization may have come from the same magma as the dyke, but the suggestion of pitchblende occurring in a fault displacing the dyke, as mentioned above, points to a younger age for the mineral deposit. It would be of interest to have an age determination made on a sample of the pitchblende, but sufficient unaltered mineral is not yet available.

The relationship to the dyke makes the walls of this and other dykes in the vicinity the most obvious place for further prospecting but, presumably, any fractures, faults, or shear zones that were open at the time of mineralization could contain pitchblende, irrespective of the presence of dykes.

Conclusions: Much work will be required to prove the merits of the property, but the discovery is of great importance because it establishes the occurrence of pitchblende in the eastern part of the pre-Cambrian Shield, at a time when the report of the "coracite" discovery had become almost legendary; and because analogous geological conditions occur over a large part of the district, hence other and perhaps more important discoveries may be made.

Hydrothermal Deposits in British Columbia

Three uranium deposits of the hydrothermal type were discovered in British Columbia in 1948 by testing properties known to contain considerable quantities of cobalt minerals. The discoveries were made at the Gem and the nearby Jewel property, in Bridge River district,[17] and at the Hazelton View property near Hazelton.[18] They show striking similarities: all consist of gold-bearing arsenopyrite associated with cobalt minerals, occurring in granodiorite, all had previously received underground exploration as gold properties, and they contain the largest amounts of cobalt minerals known in the province.[19] The economic possibilities of the discoveries are not yet known.

"Giant Quartz Vein" Deposits

The so-called "giant quartz veins" that are conspicuous features of the part of the pre-Cambrian Shield between Great Bear and Great Slave lakes are large quartz stockworks consisting of networks of quartz stringers with the intervening rock commonly replaced by silica. Many of them occur in northeasterly-trending faults, are several hundred feet wide, and are traceable for several miles. Many of these "veins" have been reopened by fracturing, and mineralized by later quartz, hematite, and small amounts of pitchblende. In some of these deposits the pitchblende is too sparse or too scattered to be of interest, but others are being developed in the hope that minable ore bodies may be outlined. If these efforts are successful, numerous other giant quartz veins that may be prospected will be found indicated on geological maps.

A large giant quartz vein lying between Hottah and Beaverlodge lakes, 200 miles north of Yellowknife, has been known for several years to contain scattered pods of hematite and pitchblende. Part of it was acquired recently by Gold-Uranium Mines Limited, who did considerable diamond drilling.

Another deposit of this type, known as the Giauque, lies 97 miles northwest of Yellowknife. When examined in 1948, trenches blasted at intervals into the rock had exposed many small fractures mineralized with hematite and pitchblende. These fractures formed a general zone perhaps 25 ft wide, extending down the center of the vein for a length of 2200 ft. Other discoveries were reported to have been made later in the year. Fairly high uranium assays can be obtained from the material in the fractures,

but these are so erratically distributed that samples taken across significant widths are low. Much systematic sampling will be required to outline the better mineralized zones and to estimate their average grade. If a satisfactory treatment process should be available, it is possible that a deposit such as this might be workable as a large, low-grade operation now or at some future time.

Sedimentary Deposits

Sedimentary strata of different ages, containing relatively small amounts of uranium, are known to occur in several parts of Canada, but most of them have not as yet been investigated in detail because of the much greater interest taken in pitchblende deposits. A small amount of uranium has recently been reported by T. L. Tanton to occur in nodules in the Sibley Series of the pre-Cambrian Shield in Ontario.[20]

The sedimentary deposit that has received the greatest attention in Canada is known as the McLean Bay occurrence, at the southeastern shore of Great Slave Lake. Here the Eldorado company has done considerable trenching, diamond drilling, and sampling. The claims are underlain by steeply-dipping beds of Proterozoic dolomite and quartzite. A zone of dolomite that displays concentric structure of possible algal origin is stained with hematite and carries low values in thorium and uranium, the ratio of the two elements being about 6 to 1. Efforts have been made to determine the thorium and uranium-bearing minerals but they have not yet been successful.

The deposit is not regarded as favorably as most Canadian pitchblende occurrences.

Secondary Deposits

Canadian occurrences of secondary uranium minerals consist of relatively small quantities of such minerals in the near-surface parts of primary uranium deposits; as coatings on the walls of mine workings and on rock in mine dumps; and, occasionally, in gravel or sand near outcrops of primary deposits, from which uranium was leached by surface water and deposited in nearby overburden. Commercial deposits of secondary uranium minerals are not thought likely to occur in Canada. The bright colors of these minerals are useful in attracting attention to the presence of primary deposits.

Thorium Deposits

Thorium deposits of present commercial importance are not known to occur in Canada, but several discoveries, of different types, have been made.

Numerous pegmatite deposits in different parts of the country are known to contain minor amounts of thorium-bearing minerals.[21]

Veins consisting of carbonate, quartz, and hematite, of the hydrothermal type, and containing appreciable quantities of thorium in a form as yet unidentified, were discovered in 1948 at the northern edge of the pre-Cambrian Shield near the railway that extends from Cochrane, Ontario, to Hudson Bay.[22] Some geologists who have examined these discoveries believe that the age of the mineralization may be later than Middle Devonian.

The occurrence of considerable thorium in pre-Cambrian dolomite at Great Slave Lake has already been mentioned in the section of this paper dealing with sedimentary uranium deposits.

Several samples of placer concentrates from British Columbia and the Northwest Territories have proved radioactive. Whenever exhaustive tests were made the radioactivity was shown to be due to the presence of thorium, not uranium. This is to be expected, because of the greater stability of thorium minerals. The thorium minerals concerned were either known or suspected to be monazite. The only thorium-bearing placer as yet examined in detail proved to be too small to be of present economic interest.

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Quantitative Efficiency of Separation of

Coal Cleaning Equipment

by W. W. Anderson

A formula for quantitative efficiency is proposed, in which the efficiency value is a function of the improperly distributed material at the gravity of separation effected by the cleaning equipment. The misplaced material is expressed as a percentage of the total material discharged from the equipment. Numerous examples are given, demonstrating application of the formula to various types of separation, including multiple-product separations.

WEBSTER'S dictionary gives the following definition for "efficiency": "Effective operation as measured by a comparison of actual and possible results."

Engineers think of this definition in terms of the ratio of output to input, and since they learn early in their careers that output cannot be greater than input, they usually consider a machine to be 100 pct efficient if it operates perfectly, and to be 0 pct efficient if it performs no function at all. Hence, values of efficiency between 0 and 100 pct reflect how well or how poorly a machine has performed; and any method of calculation for determining efficiency that does not show 0 pct when no function has been performed, and 100 pct when a perfect job has been done, is looked upon with skepticism.

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The literature on performance of coal cleaning equipment contains many references to efficiency of performance, but there is no unanimous agreement among the authors as to how the efficiency should be determined. Numerous methods of calculation have been proposed from time to time, most of which attempt to fulfill the definition of efficiency by comparing output to input, using float and sink data or some other characteristic, such as dry ash content, as a measure. However, to the best of our knowledge, none of the methods advocated in the past meets the requirements of a practical engineer in judging cleaning plant performance, because the formulas previously advocated do not confine numerical values of efficiency between 0 and 100 pct, and the literature does not contain an explanation of the meaning of the formulas under various conditions of operation.

Therefore, the following formula, based on Yancey's concept of misplaced material,[1] is proposed as a measure of efficiency which will conform with the requirement that the values of efficiency must be confined between 0 and 100 pct:

$$\text{Quantitative Efficiency} = 100.0 - (\text{pct wt sink}^* \text{ in cleaned coal} + \text{pct wt float}^* \text{ in refuse})^\dagger$$

* At gravity of separation effected by cleaning equipment.
† Total product basis of the size of material under study.

It will be noted from this formula that an actual separation between cleaned coal and refuse must be effected by the cleaning equipment before the formula can be used; also, that improperly distributed material in the products must be determined at the gravity of separation, rather than at some arbitrary test gravity.

It will further be noted from this formula that the weight of cleaned coal recovered and the weight of refuse rejected by the equipment have their proper influence, because the sink in the cleaned coal and the float in the refuse are expressed in the equation as a per cent weight of the total products of a definite size discharged from the plant.

This formula has one other important characteristic, in that it requires no knowledge of the feed to the cleaning unit. Comparison of data on the products from a cleaning unit with the feed to the unit can lead to many worthwhile conclusions, but not in regard to the efficiency of separation effected by the equipment. The reason for this is that mechanical fracture of many pieces of the feed occurs during the cleaning process; and in some processes crushers are deliberately incorporated in the flow to produce size degradation to free impurities from cleaned coal, thereby altering the washability characteristics of the material while in process.

When such size degradation occurs in a cleaning process, it is quite possible that this crushing will result in a reduction in ash content in the finer sizes of cleaned coal when compared with the raw coal feed, even though no true separation has been effected. This is due to the fact that the finer sizes have been "sweetened" by the crushing of cleaned coal from some coarser size. Many formulas advocated in the past show high cleaning efficiencies

when a condition such as this occurs, erroneously crediting the effects of crushing to cleaning efficiency.

It is our opinion that size degradation occurs in the handling of all materials through a cleaning plant, and for this reason efficiency values must be calculated in terms of the material discharged from the plant, rather than in terms of feed to the plant. The formula proposed herein is based on this belief.

The concept of efficiency can best be understood by examination of a few sets of data. There are essentially three types, as follows: (1) no true separation and no efficiency, (2) true separation at 100 pct efficiency, and (3) true separation at an efficiency less than 100 pct.

Type 1, No True Separation and No Efficiency: If for some reason or other a material should pass through a cleaning unit and be divided into a cleaned coal product and a refuse product, each having the same gravity consist after discharge from the machine, a set of data would result such as is shown in table I.

Table I. Specific Gravity Consist of Products, Type 1 Separation

Specific Gravity		Cleaned Coal	Refuse
Sink	Float	Pct Wt	Pct Wt
	1.30	55.0	55.0
1.30	1.40	8.0	8.0
1.40	1.50	3.0	3.0
1.50	1.60	2.0	2.0
1.60	1.70	1.5	1.5
1.70	1.80	1.5	1.5
1.80	2.00	2.0	2.0
2.00		27.0	27.0
		100.0	100.0

In this case there has been a division of the feed into two products, but it is obvious that no true separation in accordance with specific gravity has been effected. Therefore, there can be no efficiency of separation.

Type 2, True Separation at 100 pct Efficiency: The opposite extreme from no separation at all is perfect separation, such as is exemplified in table II.

Table II. Specific Gravity Consist of Products, Type 2 Separation

Specific Gravity		Cleaned Coal	Refuse
Sink	Float	Pct Wt	Pct Wt
	1.30	90.0	0.0
1.30	1.40	5.0	0.0
1.40	1.50	4.0	0.0
1.50	1.60	1.0	0.0
1.60	1.70	0.0	1.0
1.70	1.80	0.0	4.0
1.80	2.00	0.0	5.0
2.00		0.0	90.0
		100.0	100.0

The foregoing figures demonstrate an effective separation at a specific gravity of 1.60 and at an efficiency of 100 pct, because there is no sink material in the cleaned coal at a gravity greater than 1.60, and there is no float in the refuse at a gravity less than 1.60.

Type 3, True Separation at an Efficiency Less Than 100 pct: Types 1 and 2 separations cover extreme conditions, and the simple sets of data presented to exemplify them require no extensive calculations to determine their meaning. However, the significance of data exemplifying a Type 3 separation is not so obvious, despite the fact that simple sets of data can be presented of Type 3 separations. Therefore, the several sets of data presented hereinafter of Type 3 separations must be examined in greater detail; but prior to this examination it seems advisable to point out that Type 3 separations are of two kinds: (*a*) efficiencies above 50 pct, wherein the cleaned coal contains more float than the refuse; and (*b*) efficiencies below 50 pct, wherein the refuse contains more float than the cleaned coal.

These two kinds of Type 3 separation will now be considered.

(*a*) *Efficiencies Above 50 pct:* The values in table III are typical of a very high order of efficiency, approaching perfect separation.

Table III. Specific Gravity Consist of Products, High Order of Type 3 (*a*) Separation

Specific Gravity		Cleaned Coal	Refuse
Sink	Float	Pct Wt	Pct Wt
	1.30	90.0	0.0
1.30	1.40	4.0	0.0
1.40	1.50	3.0	0.0
1.50	1.60	2.0	1.0
1.60	1.70	1.0	2.0
1.70	1.80	0.0	3.0
1.80	2.00	0.0	4.0
2.00		0.0	90.0
		100.0	100.0

It is apparent from these gravity consists that a true separation has occurred between the cleaned coal and the refuse products, and that the separation has been effected in the gravity range of 1.50 to 1.70. It is also apparent that the separation has not been perfect, because there has been an overlapping of material in the gravities between 1.50 and 1.70. But the exact gravity of separation and the efficiency of separation are not evident.

However, if we assume that the total products are divided equally between cleaned coal and refuse, it will be seen that the gravity of separation occurred at 1.60. Then, the quantitative efficiency of separation at this gravity can be determined to be 99 pct, as follows:

		Pct Weight Misplaced
Cleaned Coal	= 50 pct of total products, and contains 1 pct sink at 1.60 gravity	0.50
Refuse	= 50 pct of total products, and contains 1 pct float at 1.60 gravity	0.50
	Total misplaced	1.00

Quantitative efficiency = 100 — 1 = 99 pct

In contrast to a nearly perfect separation, there is the type of separation which is so poor that the

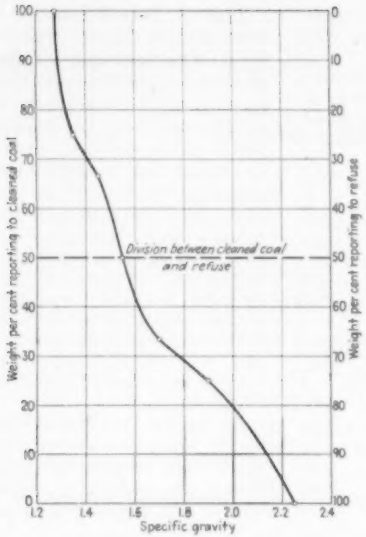

Fig. 1—Distribution of total weight between cleaned coal and refuse.

efficiency is of a low order, approaching 50 pct. For example, consider the figures in table IV.

Table IV. Specific Gravity Consist of Products, Low Order of Type 3 *(a)* Separation

Cleaned coal = 50 pct of combined products
Refuse = 50 pct of combined products

Specific Gravity		Cleaned Coal	Refuse
Sink	Float	Pct Wt	Pct Wt
	1.30	1.0	0.0
1.30	1.40	6.0	2.0
1.40	1.50	10.0	5.0
1.50	1.60	76.0	76.0
1.60	1.80	5.0	10.0
1.80	2.00	2.0	6.0
2.00		0.0	1.0
		100.0	100.0

The weight distribution in the various gravity fractions of the cleaned coal and refuse is shown in table V.

Table V. Distribution of Material, Low Order of Type 3 *(a)* Separation

Specific Gravity		Pct Wt of Fraction	
Sink	Float	Cleaned Coal	Refuse
	1.30	100.0	0.0
1.30	1.40	75.0	25.0
1.40	1.50	66.7	33.3
1.50	1.60	50.0	50.0
1.60	1.80	33.3	66.7
1.80	2.00	25.0	75.0
2.00		0.0	100.0

These values indicate that an actual separation has been made between the cleaned coal and refuse, and one might suspect that this separation occurred in the gravity range of 1.50 to 1.60, although the exact gravity of separation may not be clear to everyone. Therefore, the values of the weight distribution between cleaned coal and refuse have been plotted in fig. 1.

Examination of this curve of separation shows that the curve crosses the 50 pct weight line at a specific gravity of 1.55. This means that all of the material with a specific gravity of 1.55 has been divided equally between cleaned coal and refuse. The material having a specific gravity less than 1.55 has reported in preponderant quantities to the cleaned coal product; whereas the material with a specific gravity greater than 1.55 has reported in preponderant amounts to the refuse product.

This brief explanation of the gravity of separation is given to demonstrate the correct method for determination of the gravity at which the material was actually separated into a cleaned coal and a refuse product. This method is explained elsewhere in greater detail.[3]

Based on the gravity of separation of 1.55, it will be seen from the data that there was 45 pct sink in the cleaned coal, and that there was also 45 pct float in the refuse. Hence, the quantitative efficiency of separation at a gravity of 1.55 can be determined to be 55 pct, as follows:

		Pct Weight Misplaced
Cleaned Coal	= 50 pct of total products, and contains 45 pct sink at 1.55 gravity	22.5
Refuse	= 50 pct of total products, and contains 45 pct float at 1.55 gravity	22.5
	Total misplaced	45.0

Quantitative efficiency = 100 — 45 = 55 pct

(b) Efficiencies Below 50 pct: If, for some ridiculous reason, a cleaning process should deliver a

refuse product containing more float than the cleaned coal product, a true separation, nevertheless, could be effected by the cleaning equipment, but the efficiencies of separation would be less than 50 pct. For instance, if the gravity consists of the cleaned coal and refuse products in the previous set of data were switched, we would have a condition in which the refuse would have more float than the cleaned coal product, and the separation would be indicated by the distribution of material shown in table VI.

Table VI. Specific Gravity Consist of Products, High Order of Type 3 (*b*) Separation

Cleaned coal = 50 pct of combined products
Refuse = 50 pct of combined products

Specific Gravity		Cleaned Coal	Refuse
Sink	Float	Pct Wt	Pct Wt
	1.30	0.0	1.0
1.30	1.40	2.0	6.0
1.40	1.50	5.0	10.0
1.50	1.60	76.0	76.0
1.60	1.80	10.0	5.0
1.80	2.00	6.0	2.0
2.00		1.0	0.0
		100.0	100.0

The weight distribution in the various gravity fractions of the cleaned coal and refuse is shown in table VII.

Table VII. Distribution of Material, High Order of Type 3 (*b*) Separation

Specific Gravity		Pct Wt of Fraction	
Sink	Float	Cleaned Coal	Refuse
	1.30	0.0	100.0
1.30	1.40	25.0	75.0
1.40	1.50	33.3	66.7
1.50	1.60	50.0	50.0
1.60	1.80	66.7	33.3
1.80	2.00	75.0	25.0
2.00		100.0	0.0

Again, these values demonstrate that a separation was made in the gravity range between 1.50 and 1.60; and if the values were plotted in a similar fashion to fig. 1, the gravity of separation effected would be found to be 1.55. In this case there was 55 pct sink in the cleaned coal and 55 pct float in the refuse at a specific gravity of 1.55; and the efficiency of separation can be calculated to be 45 pct, as follows:

		Pct Weight Misplaced
Cleaned Coal	= 50 pct of total products, and contains 55 pct sink at 1.55 gravity	27.5
Refuse	= 50 pct of total products, and contains 55 pct float at 1.55 gravity	27.5
	Total misplaced	55.0

Quantitative efficiency = 100 — 55 = 45 pct

Furthermore, an extreme example of an efficiency of separation of less than 50 pct would be possible, in which the quantitative efficiency would approach zero. For instance, consider the following conditions:

		Pct Weight Misplaced
Cleaned Coal	= 99 pct of total products, and contains 99 pct sink	98.01
Refuse	= 1 pct of total products, and contains 99 pct float	0.99
	Total misplaced	99.00

Quantitative efficiency = 100 — 99 = 1 pct

The foregoing examples demonstrate very simply every type of separation that can be encountered when dealing with separating equipment. However, in order to emphasize these basic types of separation, it is thought advisable to restate them, as follows:

Type 1. No true separation at zero efficiency.

Type 2. Perfect separation at 100 pct efficiency.

Type 3. True separation at an efficiency between zero and 100 pct.

(a) Efficiencies above 50 pct, wherein the cleaned coal contains more float than the refuse.

(b) Efficiencies below 50 pct, wherein the refuse contains more float than the cleaned coal.

Furthermore, if a machine makes a multiple separation, rather than a two-product separation, the efficiency of separation can be determined for each of the multiple separations and for the machine as a unit. Such values have significance for an engineer, because they are comparable from product to product within the machine, and can also be compared with data on other types of separating equipment working under similar or different conditions.

Again, examination of a few sets of data seems to offer the best means of explanation of the procedure for using the formula in connection with problems of multiple separation. First, in table VIII, a simple three-product separation will be considered, in which the efficiency is of a high order.

Table VIII. Specific Gravity Consist of Products, Three-product Separation

Specific Gravity		Cleaned Coal	Middlings	Refuse
Sink	Float	Pct Wt	Pct Wt	Pct Wt
	1.30	85.0	0.0	0.0
1.30	1.40	10.0	0.0	0.0
1.40	1.50	4.0	1.0	0.0
1.50	1.60	1.0	4.0	0.0
1.60	1.70	0.0	90.0	0.0
1.70	1.80	0.0	4.0	1.0
1.80	1.90	0.0	1.0	4.0
1.90	2.00	0.0	0.0	10.0
2.00		0.0	0.0	85.0
		100.0	100.0	100.0

It will be seen from the above set of figures that an actual separation occurred between the cleaned

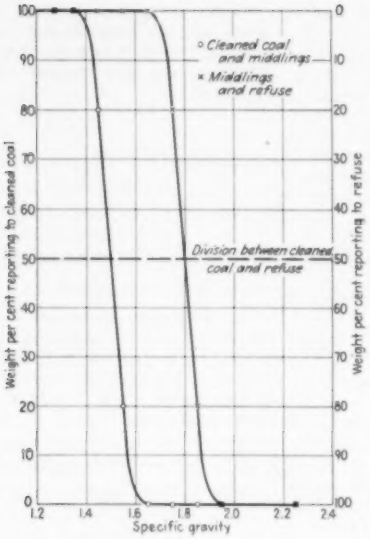

Fig. 2 — Distribution of total weight between products, three-product separation.

coal and the middlings, as well as between the middlings and the refuse, but that there was an overlapping of material in the gravity range of 1.40 to 1.60 in the cleaned coal and middlings, and another overlapping of material in the gravity range of 1.70 to 1.90 in the middlings and refuse. Hence, we know that the separation between cleaned coal and middlings occurred at some gravity between 1.40 and 1.60, and we also know that the separation between middlings and refuse occurred at some gravity between 1.70 and 1.90.

However, in each case the gravity of separation must be determined before the efficiency of separation can be determined. Therefore, in order to make the problem a simple one, it can be assumed, for purposes of explanation, that the total products are divided in equal proportion between cleaned coal, middlings, and refuse. On this basis, the weight distribution between cleaned coal and middlings is calculated to be as shown in table IX.

Table IX. Distribution of Material between Cleaned Coal and Middlings, Three-product Separation

Specific Gravity		Pct Weight of Fraction	
Sink	Float	Cleaned Coal	Middlings
	1.30	100.0	0.0
1.30	1.40	100.0	0.0
1.40	1.50	80.0	20.0
1.50	1.60	20.0	80.0
1.60	1.70	0.0	100.0
1.70	1.80	0.0	100.0
1.80	1.90	0.0	100.0
1.90	2.00	0.0	100.0
2.00		0.0	100.0

In similar fashion, the weight distribution between middlings and refuse can be calculated, obtaining the values shown in table X.

Both sets of values of distribution can be plotted on one chart, as shown in fig. 2, and it will be seen

Table X. Distribution of Material between Middlings and Refuse, Three-product Separation

Specific Gravity		Pct Weight of Fraction	
Sink	Float	Middlings	Refuse
	1.30	100.0	0.0
1.30	1.40	100.0	0.0
1.40	1.50	100.0	0.0
1.50	1.60	100.0	0.0
1.60	1.70	100.0	0.0
1.70	1.80	80.0	20.0
1.80	1.90	20.0	80.0
1.90	2.00	0.0	100.0
2.00		0.0	100.0

that the gravity of separation between cleaned coal and middlings was at 1.50, and the point of separation between middlings and refuse was at 1.80.

At a specific gravity of 1.50 there was 1 pct sink in the cleaned coal and 1 pct float in the middlings; and at a specific gravity of 1.80 there was 1 pct sink in the middlings and 1 pct float in the refuse. Thus, the efficiencies of separation can be calculated as follows:

		Pct of Total Products Misplaced
Cleaned coal	= One-third of total products and contains 1 pct sink at 1.50 gravity	0.33
Middlings	= One-third of total products and contains 1 pct float at 1.50 gravity	0.33
Refuse	= One-third of total products and contains no float at 1.50 gravity	0.00
	Total misplaced	0.66

Therefore, the quantitative efficiency of separation between cleaned coal and middlings = 100 — 0.66 = 99 1/3 pct.

Similarly,

		Pct of Total Products Misplaced
Cleaned coal	= One-third of total products and contains no sink at 1.80 gravity	0.00
Middlings	= One-third of total products and contains 1 pct sink at 1.80 gravity	0.33
Refuse	= One-third of total products and contains 1 pct float at 1.80 gravity	0.33
	Total misplaced	0.66

Therefore, the quantitative efficiency of separation between middlings and refuse = 100 — 0.66 = 99 1/3 pct.

Thus, we find that the efficiency of separation between the cleaned coal and the middlings, and between the middlings and the refuse in each case was 99 1/3 pct.

The foregoing example of a three-product separation is an extremely simple one because there was no overlapping of material in more than two products. Therefore, another example of multiple separation should be considered, in which overlapping of material occurs in more than two of the products. The figures in table XI show such a condition.

Table XI. Specific Gravity Consist of Products, Four-product Separation

Specific Gravity		25 Pct	25 Pct	25 Pct	25 Pct
Sink	Float	Cleaned Coal	No. 1 Middlings	No. 2 Middlings	Refuse
	1.30	60.0	1.0	0.0	0.0
1.30	1.40	25.0	4.0	0.0	0.0
1.40	1.50	10.0	10.0	1.0	0.0
1.50	1.60	4.0	70.0	4.0	0.0
1.60	1.70	1.0	10.0	10.0	1.0
1.70	1.80	0.0	4.0	70.0	4.0
1.80	1.90	0.0	1.0	10.0	10.0
1.90	2.00	0.0	0.0	4.0	25.0
2.00		0.0	0.0	1.0	60.0
		100.0	100.0	100.0	100.0

In the foregoing table of gravity consists an overlapping of material occurs in three of the four products in the gravity range between 1.40 and 1.90. When such a condition exists, it is not possible to determine the correct gravity of separation by examination of only two of the products, and in such case all products must be considered. For instance, in order to determine the gravity of separation between the cleaned coal and the No. 1 middlings, it is necessary to calculate the weight distribution of the material in all of the gravity fractions as it occurs in the cleaned coal in contrast to the combined value of the No. 1 middlings, the No. 2 middlings, and the refuse.

In other words, for purposes of analysis of the float and sink results in terms of the gravity of separation at which the cleaned coal was separated from the remainder of the material, the No. 1 middlings, the No. 2 middlings, and the refuse must be considered as one product, just as though only a two-product separation had been made.

Likewise, in order to determine the gravity of separation between the No. 1 middlings and the No. 2 middlings, the cleaned coal and the No. 1 middlings must be considered as one product, and compared with the No. 2 middlings and the refuse combined as one product.

This procedure of combining products must always be followed whenever there is overlapping of material into many products, because the gravity of separation is determined in terms of the distribution of the total weight of material in the gravity consist; and if the analysis of data is made excluding a portion of the overlapping material, the gravity of separation so determined will be incorrect. Furthermore, the quantitative efficiency is based on the total weight of material in the products, and all material must be accounted for if the correct efficiency is to be obtained.

Based on this procedure of combining the products properly, the weight distribution between the cleaned coal and all other products is as shown in table XII.

Table XII. Distribution of Material between Products at Cleaned Coal Discharge, Four-product Separation

Specific Gravity		Pct Weight of Fraction	
Sink	Float	Cleaned Coal	No. 1 and No. 2 Middlings and Refuse
	1.30	98.4	1.6
1.30	1.40	86.2	13.8
1.40	1.50	47.6	52.4
1.50	1.60	5.1	94.9
1.60	1.70	4.5	95.5
1.70	1.80	0.0	100.0
1.80	1.90	0.0	100.0
1.90	2.00	0.0	100.0
2.00		0.0	100.0

The weight distribution between the combined cleaned coal and No. 1 middlings, and the combined No. 2 middlings and refuse is as shown in table XIII.

Table XIII. Distribution of Material between Products at No. 2 Middlings Discharge, Four-product Separation

Specific Gravity		Pct Weight of Fraction	
Sink	Float	Cleaned Coal and No. 1 Middlings	No. 2 Middlings and Refuse
	1.30	100.0	0.0
1.30	1.40	100.0	0.0
1.40	1.50	95.2	4.8
1.50	1.60	94.9	5.1
1.60	1.70	50.0	50.0
1.70	1.80	5.1	94.9
1.80	1.90	4.8	95.2
1.90	2.00	0.0	100.0
2.00		0.0	100.0

The weight distribution between the combined

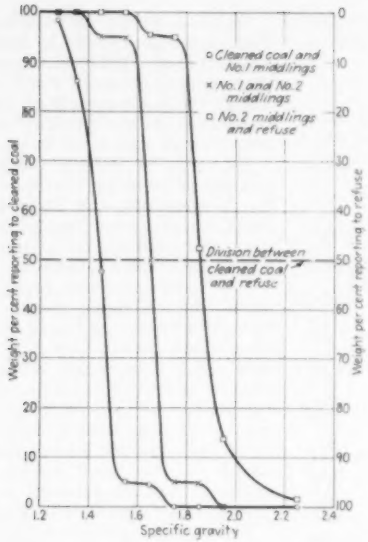
Cleaned coal and No.1 middlings
No.1 and No.2 middlings
No.2 middlings and refuse
Division between cleaned coal and refuse
Weight per cent reporting to cleaned coal
Weight per cent reporting to refuse
Specific gravity

Fig. 3—Distribution of total weight between products, four-product separation.

cleaned coal, the No. 1 and No. 2 middlings, and the refuse is as shown in table XIV.

Table XIV. Distribution of Material between Products at Refuse Discharge, Four-product Separation

Specific Gravity		Pct Weight of Fraction	
Sink	Float	Cleaned Coal No. 1 and No. 2 Middlings	Refuse
	1.30	100.0	0.0
1.30	1.40	100.0	0.0
1.40	1.50	100.0	0.0
1.50	1.60	100.0	0.0
1.60	1.70	95.5	4.5
1.70	1.80	94.9	5.1
1.80	1.90	52.4	47.6
1.90	2.00	13.8	86.2
2.00		1.6	98.4

The values of weight distribution as shown in the foregoing three tables can be plotted on one chart, as illustrated in fig. 3.

The curves in fig. 3 show gravities of separation of 1.445, 1.65, and 1.855 as the points of separation between cleaned coal and No. 1 middlings, between No. 1 and No. 2 middlings, and between No. 2 middlings and refuse, respectively. Based on these gravities of separation, and an assumption of equal weight of products, the quantitative efficiencies can be calculated as follows:

		Pct of Total Products Misplaced
Cleaned coal	= 25 pct of total products and contains 10.5 pct sink at 1.445 gravity	2.63
No. 1 middlings	= 25 pct of total products and contains 9.5 pct float at 1.445 gravity	2.37
No. 2 middlings	= 25 pct of total products and contains 0.45 pct float at 1.445 gravity	0.11
Refuse	= 25 pct of total products and contains no float at 1.445 gravity	0.00
	Total misplaced	5.11

Therefore, the quantitative efficiency of separation between cleaned coal and No. 1 middlings = 100 — 5.11 = 94.89 pct.

		Pct of Total Products Misplaced
Cleaned coal	= 25 pct of total products and contains 0.5 pct sink at 1.65 gravity	0.12
No. 1 middlings	= 25 pct of total products and contains 10.0 pct sink at 1.65 gravity	2.50
No. 2 middlings	= 25 pct of total products and contains 10.0 pct float at 1.65 gravity	2.50
Refuse	= 25 pct of total products and contains 0.5 pct float at 1.65 gravity	0.13
	Total misplaced	5.25

Therefore, the quantitative efficiency of separation between No. 1 and No. 2 middlings = 100 — 5.25 = 94.75 pct.

		Pct of Total Products Misplaced
Cleaned coal	= 25 pct of total products and contains no sink at 1.855 gravity	0.00
No. 1 middlings	= 25 pct of total products and contains 0.45 pct sink at 1.855 gravity	0.11
No. 2 middlings	= 25 pct of total products and contains 9.5 pct sink at	

	1.855 gravity	2.37
Refuse	= 25 pct of total products and contains 10.5 pct float at 1.855 gravity	2.63
	Total misplaced	5.11

Therefore, the quantitative efficiency of separation between No. 2 middlings and refuse = 100 — 5.11 = 94.89 pct.

Consideration will now be given to the combined, or overall efficiency of a multiple-product machine, the operation of which is inherently more difficult to understand than that of a two-product machine. For instance, in the preceding example, four products were obtained at three points of separation, and the combined efficiency is not evident. However, if this equipment had been operated to produce a two-product separation, making a cleaned coal at a specific gravity of separation of 1.445, and the two middlings products had been combined with the refuse as a total refuse product, the efficiency of the machine would have been 94.89 pct.

On the other hand, if the No. 1 middlings had been combined with the cleaned coal, and the No. 2 middlings with the refuse, producing a two-product separation at a specific gravity of 1.65, the efficiency would have been 94.75 pct.

Similarly, the efficiency of separation would have been 94.89 pct if the No. 1 and No. 2 middlings had been combined with the cleaned coal to produce a two-product separation at a specific gravity of 1.855.

However, if none of the four products had been recombined, the overall efficiency of the machine would have been the average of the efficiencies at the three points of separation, namely, 94.84 pct. The overall efficiency must be the average efficiency, because the efficiencies of separation at each point of separation are determined on a proportional basis; hence, the average of the efficiencies at each point of separation can be calculated on a straight arithmetic basis.

Thus, it will be seen that the overall efficiency of any equipment must be considered from the standpoint of the final disposition of the products. In this respect, it should be noted that there are a number of multiple-product machines that are operated to produce numerous products, which are combined into fewer products after removal from the machine. The Rheolaveur washer, air jigs, and water jigs are typical of such equipment; and it is the final combination of products which should be examined to determine the combined efficiency of the units, because combining products automatically reduces the final number of separations and products. This does not mean that study of the separations at intermediate points of separation is not valuable, because many times such study leads to important conclusions in regard to the effect of the intermediate separations on the final products.

Another example explaining why the final products must determine the overall efficiency of any machine is the concentrating table. It is a well-known fact that concentrating tables produce their final efficient separation of products by continuously effecting inefficient separations at each riffle on the deck of the table; and, strange as it may seem, the net effect of these many inefficient separations at each riffle finally results in a highly efficient separation at or near the corner of the table where the final two-product separation is usually made.

Actually, the reason why the cumulative effect of many inefficient separations is a final separation of a high order of efficiency is because the separation made at each riffle is made on a material different from that treated at the previous riffle; and it is this continuous retreatment which finally results in an efficient separation at the discharge side of the table.

In similar manner, a jig discards a refuse product by means of the No. 1 elevator, and the products discharged at the succeeding elevators are of a different character because the material in the jig is continuously retreated between each successive elevator. The efficiency of separation of the materials at each elevator may not be perfect; nevertheless, the final separation between the combined elevator products and the cleaned coal overflow may be of a highly efficient order.

There are numerous other types of cleaning equipment which likewise continuously extract portions of the feed, and continue to retreat the remainder of the feed before the final separation is completed.

Therefore, it is obvious that inefficiencies at any one point of separation in a unit should not be cause for condemning a piece of equipment as inefficient if the net result of combining products effects a final efficient separation. Too often a machine is termed inefficient because there has been a misunderstanding in regard to the true conditions of separation which actually took place. The important point to be stressed is that frequently, when a satisfactory cleaned coal product is not obtained, the cleaning equipment is incorrectly labeled as being inefficient; whereas many times the actual separation is quite efficient, but the equipment is operating at too high a gravity to obtain a satisfactory analysis in the cleaned coal product.

Finally, a word of caution should be included to warn against the prevalent practice of confusing recovery of products with efficiency of separation. Recovery and efficiency are two entirely separate measures of cleaning performance, as will now be explained.

For instance, using the preceding example, if a two-product separation had been made, and the two middlings products had been discarded with the refuse by some piece of cleaning equipment that was able to make a nearly perfect separation at a specific gravity of 1.445, the efficiency of separation would have been about 5 pct higher than that of the machine that made this four-product separation.

On the other hand, the total recovery of material might very well have been much less than was accomplished by the four-product machine. The reason for this is that the total refuse product from a two-product machine would have included very nearly all of the feed having a gravity higher than 1.45; whereas the four-product machine classified a fairly clean coal and a fairly clean refuse, as well as two middlings products, which could be crushed

and retreated for further recovery, or sold to a different market.

Hence, despite the fact that the overall efficiency of the four-product machine in this example was only about 95 pct, the total recovery of marketable material could be greater than that produced by a more efficient two-product machine. It is true, of course, that greater recovery could be accomplished by installing several very efficient two-product machines in series, or by installing some other multiple-product machine of higher efficiency; but that is a question of economics, and will not be discussed in this paper.

To emphasize the difference between recovery and efficiency, the preceding example demonstrates that the recovery could be increased from 25 to 50 pct by accepting a washing gravity on a two-product separation of 1.65 instead of 1.445; and the efficiency of separation at 1.65 gravity will be 94.75 pct, in comparison with 94.89 pct at 1.445 gravity. This is a tremendous difference in recovery, but an almost negligible difference in efficiency. Of course, the ash contents of the products will not be the same in both cases, but that does not alter the fact that the recoveries are decidedly different, while the efficiencies are practically the same.

Furthermore, the preceding example should not give the impression that multiple-product machines are necessarily less efficient than two-product machines. No inferences should be derived from the examples of data in this paper regarding the relative merits of various types of equipment, because the efficiency of operation of any machine is dependent on the conditions under which the machine is operated. Conclusions in regard to the relative merits of various types of equipment can be deduced only from comparable sets of data obtained on various pieces of equipment when operating with the same type of feed under similar conditions.

Nevertheless, it may be thought that since efficiencies determined by the recovery method are obtained by comparing the actual recovery of cleaned coal with the theoretical recovery as shown by the cleaned coal ash content compared with the feed data, the efficiency by this method is a true measure of cleaning performance. Such a comparison would at first appear valid, but further examination of this method develops four reasons why it is misleading and should not be used:

1. The recovery method of determining efficiency credits effects of crushing to cleaning efficiency.

2. Efficiency values greater than 100 pct can be obtained by use of the recovery method of calculation, and the significance of an efficiency greater than 100 pct has never been explained.

3. The recovery method ignores the quality of the refuse, although the method should be just as valid for the refuse as for the cleaned coal.

4. The use of the method for a problem in multiple separation has never been explained; hence, the method is not generally useful for all types of separation.

In conclusion, it should be pointed out that some aspects of the concept of improperly distributed material have been discussed previously. The literature on the subject seems to indicate the following history of events:

In 1912, David Hancock proposed the Hancock chart,[2] which was probably the beginning of a basic understanding of the separation effected by a washer.

In 1929, Yancey and Fraser[3] discussed efficiencies in relation to the Hancock chart, and incorrectly labeled recoveries as efficiencies.

In 1937, K. F. Tromp[4] suggested that the curve of improper distribution resembled the Gaussian error distribution curve familar to statisticians.

In 1938, Yancey[1] and his associates discussed improperly distributed material, but concluded that it did not give a satisfactory measure of efficiency because it did not take into account the relative difficulty of separation.

In 1947, Driessen[5] proposed a "relative-error curve" for measuring the faults of washing, but did not convert his values of error area to actual values of efficiency.

The formula for efficiency proposed herein is the result of following the work of previous investigators to its logical conclusion, namely, that a perfect separation, in which there is no improperly distributed material, constitutes a 100 pct efficient separation, and that anything less than a perfect separation can be expressed by 100 pct minus the improperly distributed material at the gravity of separation.

This formula does not attempt to take into account the various factors, such as size consist and size range of feed, gravity consist of feed, rate of feed, and other conditions that influence the efficiency of separation. It is acknowledged that the efficiency of separation is influenced by the conditions under which a machine must operate, but the numerical value of the efficiency should not have to express all of the combinations of operating conditions which caused its result. The operating conditions may be the cause for the result, but they are not the result, and should not be confused therewith.

Much of the above discussion is based on material that is not new in itself, but it is hoped that this re-examination of an old problem offers a practical approach to a better understanding of the relationships between efficiencies and gravities of separation, and that the procedure for analyzing problems of multiple separation will clarify a blind spot in the literature on coal cleaning.

As stated previously, this formula was motivated by the need for a practical means for judging cleaning plant performance, and this paper has resulted from our experience in its use over a period of four years. During this time numerous types of equipment and cleaning plants have been tested and evaluated with various coals; and these tests and evaluations have developed patterns of separation that could not be found in the literature on coal cleaning.

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Some Factors Influencing the Performance of

Single Retort Underfeed Stokers

by H. A. Baumann,

T. S. Spicer, and

C. C. Wright

Experimental data are presented showing the influence of size consist and firing rate upon the performance of bituminous coal-fired, single-retort, industrial underfeed stokers. Size segregation, degradation, and distributor data are also included. The conclusion is reached that size consist of the fuel as fired, is a critical factor in its performance on this class of equipment.

ALTHOUGH the efficiency of coal utilization has improved steadily since the turn of the century as a result of continued research and development, little of this increase in efficiency can be credited to improvements in the single retort stoker. There are still numerous problems connected with the burning of coal on this type of unit, the solution of which would be a stimulus to coal and stoker sales and a boon to the user. Among the more critical of these problems is that of burning slack bituminous coal of strong caking tendency and containing large percentages of fine material. With the increased trend toward mechanical mining, it may be expected that in the future an even larger percentage of the coal available for use on this class of stokers will contain higher percentages of the fine sizes, thus aggravating an already serious problem.

During the war, the problem of efficient and economic burning of available coals on the single retort industrial stoker, particularly the class 4 stokers (301 to 1200 lb per hr), became quite critical in many localities. In addition to the inherent problems associated with the use of this class stoker, the coals available were frequently not those best suited for use on this equipment and the manpower shortage made it difficult as well as expensive for the consumer to provide the labor necessary to keep the fires in reasonably satisfactory condition. Although exact figures on the number of class 4 stokers are not available, the yearly tonnages of coal consumed are of considerable consequence and have been estimated* at between 15 and 20 million tons. Largely as a result of these considerations, the stoker research staff of the Mineral Industries Experiment Station of The Pennsylvania State College was requested to study the performance of the single retort industrial stoker and to undertake a program of research on the more efficient burning of bituminous coal on this class of unit. Various aspects of overall problem have been investigated since the inception of this program in 1942, only one of which is here reported.

* E. G. Bailey: Trends of Coal Preparation and Fuel Burning Equipment. *Proc.* Fuel Engineering Conference, Coal Bureau, Upper Monongahela Valley Association, July 24, 1942.

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General Considerations

There are numerous factors which appear directly or indirectly to affect the overall performance of bituminous coal underfeed stokers, such as the caking or matting properties of the coal, the contact relationships between coal and air, the ash fusi n characteristics, the burning rate, and the design of the stoker and furnace in which the coal is burned. An analysis of the influence of each of

these factors indicates, however, that all are interrelated and interdependent. For example, the contact relationships between coal and air has a definite bearing upon the caking and matting properties as evidenced by the phenomenon of preoxidation and its influence upon the character of the coke. Contact relationships also influence the burning rate since this reaction is a function of surface area and oxygen concentration. Similarly, the character of the coke influences the contact relationship between air and fuel, and is itself dependent upon the flow of heat to and from the fuel bed, which in turn is influenced by furnace design and by burning rate.

Reducing the problem of combustion on underfeed stokers to its essentials, it would appear that one of the primary factors governing performance is the rate of heat transfer to, from, and through the coal. For a given installation, this rate of heat transfer is largely a function of burning rate which is influenced to an appreciable extent by the size consist of the bed. It is the purpose of this paper to present data showing some of the relationships existing between performance, burning rate, and size consist of the coal in the hope that they may serve as an aid in interpreting actual performance and may point the way to design improvements.

Experimental Investigations

Equipment and Installation: The single retort underfeed stoker employed in the studies herein reported has a nominal rating of 540 lb of coal per hour. The general features of the stoker are shown schematically in fig. 1. The coal is delivered from hopper to retort by means of a screw, and distribution along the retort is accomplished with a ram type pusher; the grate bars are of the so-called movable type in which every third bar moves while the pair between is fixed.

The stoker is installed in a well-insulated brick furnace with a water pan replacing the boiler. Although a small percentage of the heat liberated is absorbed by the water in the pan, the major portion is discharged up the stack as sensible heat of the gases. This arrangement is much cheaper to install and permits greater flexibility for test purposes than does the standard stoker-boiler installation. It will be noted in fig. 2 that the front wall of the furnace is so constructed that from the turret above the hopper level it is possible to see

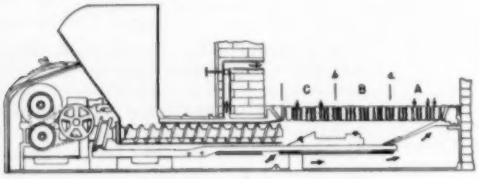

Fig. 1 — Sectional elevation of experimental stoker.

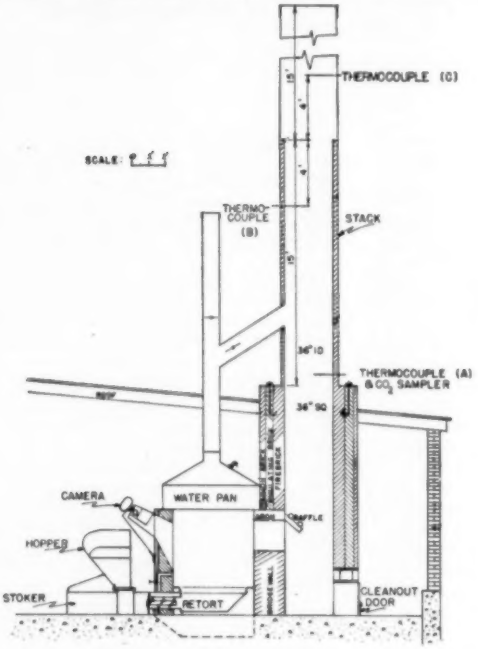

Fig. 2 — Sectional view of industrial stoker experimental equipment.

the entire fuel bed and, when desired, pictorial records of the operation can be secured.

Forced air delivered to the stoker is measured in an especially constructed inlet duct by means of a calibrated Pitot tube. The carbon dioxide content and temperature of the stack gases are recorded on a strip chart instrument. The arrangement of the air duct and the points at which air, carbon dioxide, and temperature are measured are indicated in fig. 2 and 3.

Properties of Coals Tested: *Description and Analyses:* Descriptions and analyses of the coals tested are shown in table I. Although these coals were all from the Pennsylvania coal fields, it is believed that the results and conclusions are generally applicable to bituminous coal of comparable quality from other sources.

Screen Analyses: Screen analyses for each of the coals tested are presented in graphic form in fig. 4. These analyses were made as follows: a 2000 lb sample of each of the test coals was taken from stock and each was coned and quartered according to the standard ASTM procedure. Alternate quarters were combined and the 1000 lb samples were screened using the Tyler Ty-Lab shaker with 2, 1½, 1¼, 1, ¾, and ½ in. round hole screens, and U.S. screens, No. 4, 8, and 16. The screen fractions were remixed by coning with as little handling as possible and were used in subsequent degradation tests.

Coal Degradation: Degradation tests were performed on each coal by feeding each of the previously screened and remixed samples through the stokers at a feed rate of approximately 540 lb per hr, which is the nominal rating for the unit tested. The coal fed through the stoker was collected as soon as it passed the level of the top of the grate bars. Screen analyses on the coal thus collected were performed in a manner similar to that used in the original screen analysis. The results of these degradation tests together with screen analyses of the original coals are shown in fig. 5, 6, and 7 for coals *A*, *B*, and *C*, respectively.

Table I. Description of Test Coals of Pennsylvania

Source and Size of Coals

Coal	Seam	County	Nominal Size
A	Pittsburgh	Allegheny	2 in. x 0 nut and slack
B	Lower Kittanning	Cambria	3 in. x 0 nut and slack
C	Lower Kittanning	Cambria	¾ in. x 0 slack
D	Upper Freeport	Indiana	2 in. x 0 nut and slack
*E**	Upper Freeport	Indiana	3 in. x ¾ in.

Coal Analyses, As Received Basis

	Coal *A*	Coal *B*	Coal *C*	Coal *D*
Proximate Analysis:				
Moisture, pct	1.8	0.8	1.2	0.3
Volatile matter, pct	38.3	21.1	22.7	29.8
Fixed carbon, pct	53.4	71.3	68.9	59.8
Ash, pct	6.5	6.8	7.2	10.1
Sulphur	1.62	2.2	1.0	1.5
Heating value, Btu	13,415	14,430	14,330	13,725
Ash Fusion Characteristics:				
I.D.T., °F	1,930	2,160	2,120	2,250
A.S.T., °F	2,240	2,430	2,450	2,470
F.T., °F	2,360	2,530	2,540	2,740
Bulk density, lb per cu ft	50.5	49.5	48.5	49.0
BSC Swelling Index	7.5	9+	9+	8.0

* Coal *E* was prepared from coal *D* at the mine by passing over a ¾ in. screen.

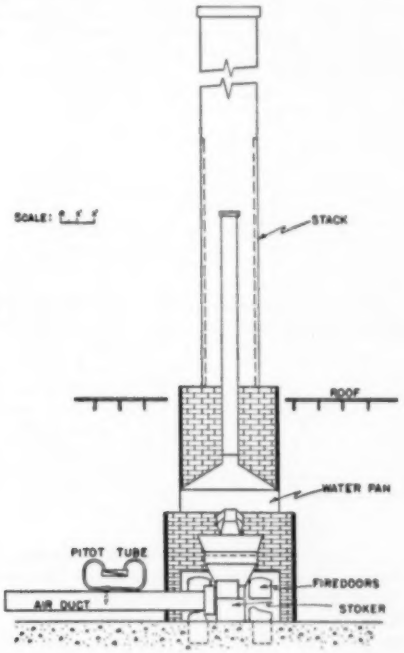

Fig. 3 — Front view of industrial stoker experimental equipment.

Coal Segregation: The coal samples used in size degradation tests were made up to the original screen size analysis by addition or subtraction of fractions of the same coals obtained from screen analysis of the alternate quarters not used in the degradation tests, and were used for segregation tests. Segregation of sizes in passing from hopper to retort of the stoker was determined by dividing the retort into three sections, A, B, and C, of equal area at the grate bar level, as illustrated in fig. 1. Plates were made which could be inserted into the retort at "a" and "b" to permit removal of the coal from the three sections without intermixing of the coal in the three sections. With the plates removed, coal was fed from the hopper until the retort was filled to the level of the top of the grate bars. The stoker was then stopped, the plates inserted at "a" and "b" and the coal from each section carefully removed and its size distribution determined by screen analysis. The plates were then removed and feeding of the coal resumed. After 30 min of feeding coal the stoker was again stopped, the plates inserted at "a" and "b," excess coal above the level of the top of grate bars removed, the coal in each of the three sections carefully removed and its size distribution determined by screen analysis. The foregoing procedure was repeated with the feeding time being continued for 60 min rather than 30 min. Fig. 8, 9, and 10 show the screen analysis of the original coal sample, and screen analyses after the first filling of the retort, after 30 min of continuous operation, and after 1 hr of continuous operation, for the coal in sections A, B, and C of the retort when using hard structure coal *A*. Fig. 11, 12, and 13 show similar data for soft structure coal *B*.

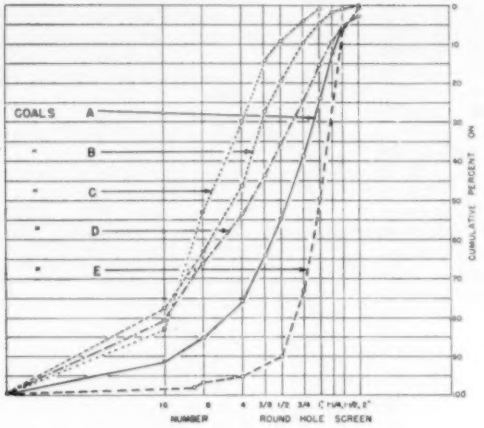

Fig. 4 — Screen analyses of test coals.

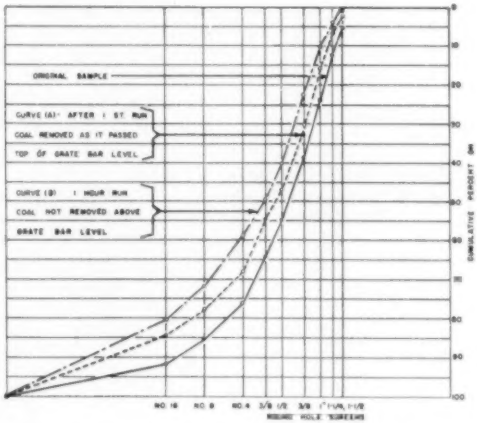
ORIGINAL SAMPLE
CURVE (A) AFTER 1 ST. RUN
COAL REMOVED AS IT PASSED
TOP OF GRATE BAR LEVEL
CURVE (B) 1 HOUR RUN
COAL NOT REMOVED ABOVE
GRATE BAR LEVEL
CUMULATIVE PERCENT ON

Fig. 5 — Screen analyses of degradation test on coal *A*.

Coal Distribution: During the operation of the stoker for segregation and degradation tests, observations were made on the quantity distribution of the various coals over the grate area. No endeavor was made to modify the distribution by changing the arrangement of the pusher blocks once an arrangement had been secured which gave good quantity distribution for the first coal tested. Photographs were made at definite times to record the quantity distribution of the coals.

Combustion Tests: A large number of combustion tests have been completed in which various factors such as shape and design of grate bar and

retort, motion of distribution ram, air zoning and related mechanical features as well as such factors as the influence of firing rate, size consist of the fuel and preoxidation of the fuel by various methods have been studied. In each of these combustion tests the following standard operating procedure has been employed.

The fire is lighted using wood and kerosene and the furnace brought up to temperature over a period of 2 to 3 hr. During this period the air supply is adjusted to approximately 50 pct excess on the basis of predicted coal feed rate and measured air to fan. The true coal feed rate is subsequently determined from the actual weight of coal fed during the test and the reported excess air is calculated accordingly. At the start of the actual test period,

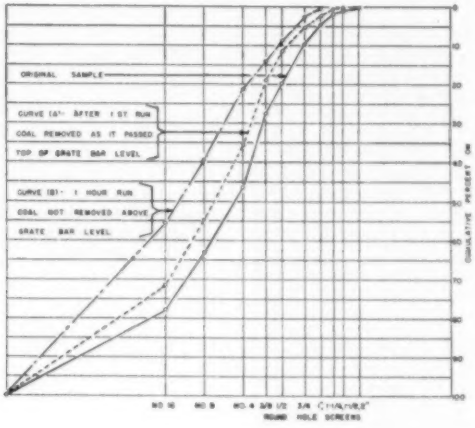

Fig. 6 — Screen analyses of degradation test on coal *B*.

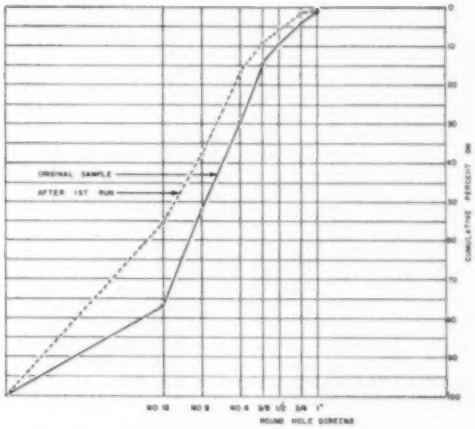

Fig. 7 — Screen analyses of degradation test on coal *C*.

Table II. Combustion Data for Firing Rate Tests

Coals	Normal Rating, Pct	Feed, Lb per Hr	Air[a]		CO_2 Pct			Stack Temp. °F.			Relative Performance
			Excess Corrected, Pct	Total Corrected CFM	Min.	Max.	Avg.	Min.	Max.	Avg.	
A	50	270	54	970	3.8	7.4	5.5	820	1,220	950	Fair
A	100	541	49	1,900	7.2	12.0	9.0	1,280	1,480	1,350	Good
A	130	787	1	1,875	6.0	12.0	10.0	1,520	1,720	1,600	Good
B	50	283	35	975	4.0	7.0	6.5	840	1,220	1,100	Poor
B	100	547	45	1,995	6.4	8.9	8.0	1,150	1,420	1,300	Fair
B	150	788	21	2,380	8.0	12.4	10.5	1,200	1,820	1,500	Fair
C	50	263	45	975	4.0	5.4	4.6	800	980	900	Very poor
C	100	543	48	1,995	5.4	9.6	7.4	1,200	1,660	1,400	Poor
C	150	778	5	2,040	8.0	14.0	11.2	1,600	1,970	1,580	Poor

[a] These data are for measured quantities of air delivered by the fan, corrected to 70°F and 29.92 in. Hg pressure and upon the weight of coal fed.

after the furnace has attained operating temperature, the bed is leveled by raking if such action is required. After this initial raking, when employed, no manual attention is given to the bed for any reason for the equivalent of one shift, or 8 hr from the time the fire was lighted. The relative performance of the coals under the various test conditions is evaluated principally by visual observation and by time-lapse motion picture studies, although

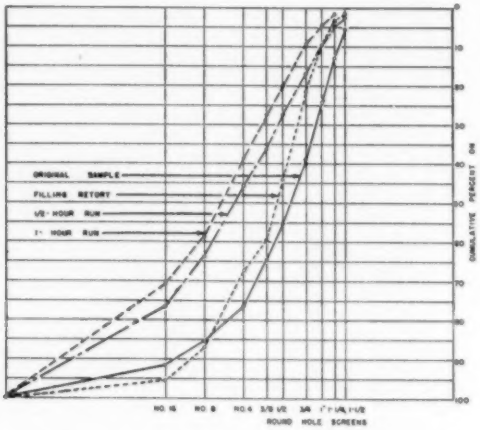

Fig. 8 — Screen analyses of segregation test on coal *A*, retort section "A."

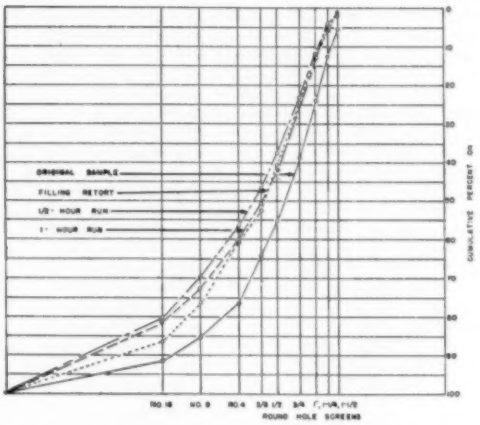

Fig. 9 — Screen analyses of segregation test on coal *A*, retort section "B."

CO_2 and stack temperature records are usually taken and serve as secondary criteria of performance.

In the present paper, the results of only two series of tests are reported because design features are beyond the scope of this report.

One series of tests was made at three different firing rates; namely, 50, 100, and 150 pct of rating for this particular stoker, in order to study the influence of this factor upon the performance of the various coals. The results are reported in table II. It should be noted that the percentage excess air reported in col. 4 is based on the volume of air delivered to the bed by the fan and the quantity of coal fed. Thus this is the theoretical excess rather than the actual, which is based on the CO_2 content of stack gases. Since air leakage was held to a minimum, CO_2 results appreciably below those corresponding to the measured excess air are an indication of the extent to which the fuel was accumulating in the bed.

The other series of tests was made in order to study the effect of size and of size consist of the fuels upon the performance. In each of these tests, the coal feed rate was maintained at approximately 100 pct of normal rating and 50 pct excess air was employed.

For this series of tests, 5000 lb of coal *D* was prepared, in which the size consist was made equivalent to that of coal *A*. Coal *A* was prepared so that 5000 lb had a size consist equivalent to that of coal *B*. Coal *D* was also modified to the extent of screening out the ¾ in. material at the mine; however, a small amount of minus ¾ in. coal remained. It would have been desirable to change the size consist of all coals to conform to each other, but this was not possible with coal *C* (¾ in. by 0) and, moreover, coal *B* had such a small percentage of the larger sizes that very large quantities would

Table III. Comparative Combustion Performance Rating of Size Consist Studies

Coals	Results
Coal *A* (2 in. x 0)	Excellent
Coal *D* with coal *A* size consist (2 in. x 0)	Good
Coal *A* with coal *B* size consist (2 in. x 0)	Fair
Coal *B* (2 in. x 0)	Fair
Coal *D* (2 in. x 0)	Poor (heavy coke masses)
Coal *C* (¾ in. x 0)	Poor (semi-coke mat)
Coal *D* (2 in. x ¾ in.)	Very poor

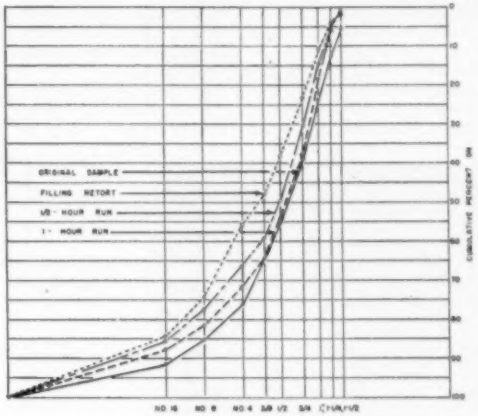

Fig. 10—Screen analyses of segregation test on coal *A*, retort section "C."

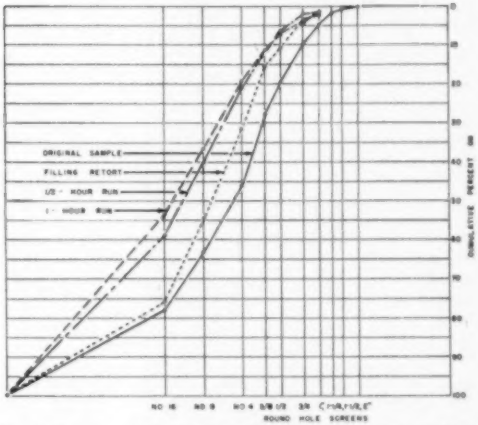

Fig. 11—Screen analyses of segregation test on coal *B*, retort section "A."

have been required. The qualitative results of these tests, as judged by fuel-bed conditions, general performance, and time-lapse motion picture studies, are presented in table III.

Discussion of Results

Degradation: During the past decade, considerable attention has been focused upon the degradation of coal in domestic underfeed stokers, but little or no mention of this phenomenon can be found in the technical literature concerned with the operation of commercial and industrial underfeed stokers. Obviously the amount of degradation will be a function of the strength characteristics of the particular coal, of the size consist of the coal, and of the design characteristics of the particular stoker. The general lack of information on this subject would suggest, however, that the importance of this factor in stoker performance may have been underrated.

The results shown in curve "A" of fig. 5, 6, and 7 for the degradation resulting when coals *A*, *B*, and *C* discharge freely at the level of the top of the retort and in curve "B" of fig. 5 and 6 where it discharges against the weight of coal above the retort formed by the natural contour of the fuel bed, suggest that degradation is a very serious consideration in some cases. It is highly probable that in actual field operation the degradation results will be worse than those shown, because in many cases the mass of coke over the retort will offer more resistance to movement of the entering coal than would the depth of coal formed by the natural contour of the unignited bed. It is apparent that with the soft structure coal *B*, appreciable degradation occurs as a result of forcing the coal upward against the head of fuel above the retort. The increase in fines (minus ⅛ in. material) is appreciable for both coals.

As may be seen from an examination of fig. 1, the stoker in which these tests were performed employed a screw feeder from hopper to retort ap-

proximately 50 in. long, which discharges the coal to the distribution ram which moves the coal along and upward in the retort. No data were secured on the relative amount of degradation produced by the screw and by the ram, but it is believed that the distributing ram produces relatively little breakage. It would be of interest to have degradation data for stokers employing only screw feed and for those employing only ram type feed. Such data do not appear to be available, however, at the present time.

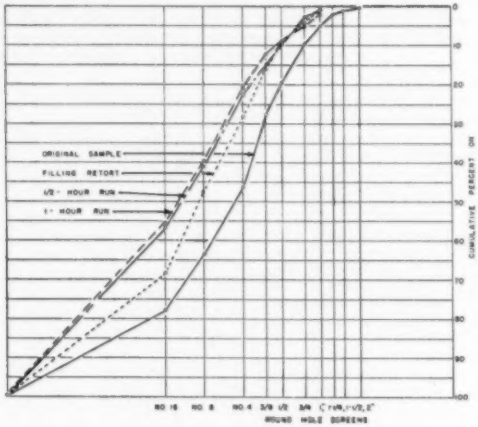

Fig. 12—Screen analyses of segregation test on coal *B*, retort section "B."

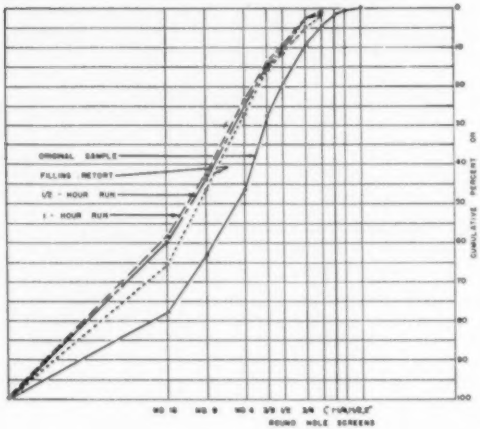

Fig. 13—Screen analyses of segregation test on coal *B*, retort section "C."

Segregation: When transporting broken solids one of the most difficult problems encountered is that of preventing segregation of the coarse material from the fines. The problem is especially difficult with the single retort underfeed stoker because the coal must be fed from the hopper into the retort and then distributed over the grate; this involves three changes in direction. The results presented in fig. 8, 9, and 10 for the distribution of coal sizes in various sections of the retort when feeding coal *A* indicate that considerable segregation may occur.

During the initial filling of the retort there is a tendency for the fines to concentrate toward the hopper end, but with continued operation when the coal is fed against a head of coal, this tendency is reversed and the fines tend to concentrate in the end away from the hopper. It was observed during these tests and recorded photographically (fig. 14) that the maximum concentration of fines occurred immediately above the point where the retort bottom changed slope, which is also the point where the end pusher block ends its stroke. Whether the change of slope or the pusher block action is the cause of this concentration cannot be stated definitely but the result of subsequent changes in pusher block arrangements would suggest that the slope is probably the controlling factor.

Similar data for coal *B* are shown in fig. 11, 12, and 13. During the initial filling of the retort the same tendency as observed with coal *A* was noted, but after the first 30 min of operation variations in distribution of sizes in the three different sections of the retort were hardly noticeable. Whether this is due to the preponderance of fines being delivered

to all sections or whether it is because the size distribution has approached some consist which minimizes segregation cannot be stated. As with coal *A*, however, it was observed and recorded photographically that there was a tendency for the fines to attain a maximum concentration in a narrow band across the retort immediately above the point where the retort bottom changed slope.

In subsequent tests, it was demonstrated that it was possible to predict with unfailing accuracy from segregation studies the location along the retort where the worst coke masses and dead spots would occur if such developed. Invariably the trouble would occur at the point of maximum fines concentration.

Distribution: As the length of retort and the grate area increase, it becomes correspondingly more difficult to maintain an even quantity distribution of coal on the grates. The problem is further complicated by the fact that the distribution is affected by size consist, moisture content, and such other physical characteristics of the coal as affect its movement. Frequently it will be found that the distribution pattern for burning beds does not follow the same pattern as so-called dry (unignited) beds, because of the segregation phenomenon discussed in the preceding section. The converse, however, is rarely true. If the distribution pattern of the dry bed is unsatisfactory, it rarely will be found to be satisfactory for the bed when burning. It is thus desirable when starting with a new coal to obtain first a satisfactory distribution with a dry bed by appropriate changes in pusher block arrangement and stroke of ram. If results with the live bed are then unsatisfactory, further changes can be made.

It should be pointed out also that the distribution pattern may be markedly different for two coals in the same equipment with the same setting of pusher blocks and stroke of ram.

Influence of Size Consist on Performance: The qualitative results presented in table III for the burning of coal *D* with initial coal *A* size consist and coal *A* with initial coal *B* size consist are indicative of the importance of this factor. It would have been more desirable to run this series of tests using coals having exactly the same size consist at the point where the coal burned, but, as previously shown, the size consist at this point is a function of several factors that are not predictable when the test is started. Consideration was given to using, as a standard, the size consist as delivered at the retort level against no head or against heads ranging up to the equivalent to that produced after 1 hr of continuous feed. In view of the uncertainty as to the exact size consist of coal *A* delivered at the grate level under actual firing conditions, however, it was decided that the safest standard to adopt would be the size consist of the original coal.

The test results established with reasonable certainty that, at a fixed firing rate in a specific installation, size consist is the most important single factor controlling the performance of the coal. That results comparable to those normally attained with coal *A* were not attained with coal *D* is believed to be due largely to the fact that the latter had a less satisfactory size consist at actual burning level.

Similar results were obtained in the test where a sample of coal *A* having the size consist of coal *B* was burned. There appeared to be little doubt that, had the consist of coal *A* been the same as that of coal *B* at the actual burning level, the results would have been essentially the same.

The results of the test with the sample of coal *D* from which most of the minus ¾ in. material had been removed is a further indication of the importance of size consist. The performance of this coal was so poor that it was impossible to maintain combustion in this particular installation. Lack of reaction surface appeared to be the cause for loss of ignition.

Influence of Firing Rate: There exists a rather common belief that as long as the stoker is oversized for its combustion assignment the performances will probably be better than if it is under-

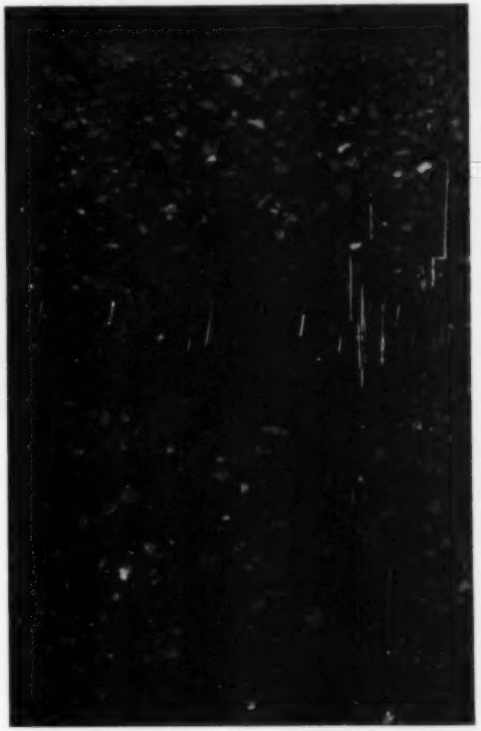

Fig. 14—View of unignited fuel bed showing band of fines concentrated above point where bottom of retort changes slope. Coal *A*. Top, bridge wall. Bottom, hopper end.

sized. That this may be an erroneous assumption is shown by the data for tests at 50, 100, and 150 pct rating presented in table II. With coal *A*, the manufacturer's rating gave the best results, 150 pct rating was almost as satisfactory and 50 pct rating gave much poorer performance. With coal *B* there was little to choose between the 100 and the 150 pct rating and had it been possible, with the installed fan, to supply sufficient excess air it is be-

lieved that 150 pct rating might have been the better of the two. The results of operation at 50 pct rating were very much poorer than at the higher ratings. The relationships between the performance for coal *C* at the three ratings were similar to those for coal *A* although performance at all ratings was very much poorer.

These results indicate the importance of the heat transfer problem. At the higher firing rates the overall furnace temperature was higher and permitted sufficient transfer of heat from furnace to fuel bed to insure at least partial coking of the coal which in turn permitted the air to channel through the coke crevices and maintain active combustion. At the low firing rate, however, insufficient heat was radiated to the bed, the character of the coking was altered and instead of producing fissured coke

Fig. 15—View of ignited bed burning coal *C* at 50 pct rating. Green coal blanketing 80 pct of total grate area. Top, bridge wall. Bottom, hopper end.

over the retort section, a relatively impervious mat of semifused coal resulted, through which the air could penetrate only with difficulty. This results in most of the air blasting along the grate bars and produces intense localized combustion around the edge of the bed with virtually no combustion over the retort section. In extreme cases, such as with coal *C*, where in addition to poor heat transfer from the furnace, the size consist is such that heat transfer through the coal itself is poor, the condition of the fuel bed may become extremely poor with green coal pushing up through the center of the bed all along the retort. This mass of green coal spills down over the bed of fused or semifused coal on the grates and may at low firing rates extinguish the fire. A close approach to this condition may be seen in fig. 15 where coal *C* was fired at 50 pct rating. Green coal blanketed the bed over approximately 80 pct of the total grate area.

Summary

Experiments with a limited number of coals in a specific stoker-furnace test installation indicate that at a given firing rate, size consist of the fuel as actually burned is one of the major controlling factors in determining the relative performance of different coals. With a given coal, performance is influenced largely by the rate of heat transfer to, from and through the fuel bed which to some extent is a function of the firing rate. The probability exists that the best size consist for producing optimum performance of a given coal in a specific installation is not the one that is most economic for the coal producer to supply. In some cases, it may be feasible for the mines to furnish a consist more suitable for use on the single retort stoker, but in general other means of overcoming the difficulty must be found.

Firing rate studies on a limited number of coals indicate that performance can in most cases be improved by judicious selection of firing rates where load and other conditions warrant such a practice. While the optimum firing rate may coincide with the manufacturer's rating, this is not necessarily the case since it is impossible for the manufacturer to predict the performance of his equipment with a wide variety of coals which behave differently. The plant engineer in cooperation with combustion engineers from the coal supplier and equipment manufacturer can do much to establish the best conditions for operation under the specific local conditions. The test results suggest that the so-called coking, caking, or matting characteristics of the coal are not intrinsic characteristics and that the performance of a number of coals, generally considered quite different in these characteristics, is actually quite similar when conditions of size consist, as fired, and of rate of heat transfer are about the same.

Although size consist and rate of heat transfer appear to be controlling factors in determining performance, it appears probable that much could be accomplished in the design of both stoker and furnace which would influence the effect of these factors and permit the efficient burning of bituminous coals which at present do not generally give satisfactory performance on this class of stoker.

Acknowledgments

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Research in Coal Geology

by Gilbert H. Cady

The present article calls attention to the activities of a committee on coal research of the sister society of Economic Geologists. The field of coal geology crosses the boundary between the two societies and joint meetings are as applicable here as in the metallic and other nonfuel fields of mining and natural resources recovery. The six fields of coal geology are briefly described and some of the ways in which they touch on coal mining, preparation, and recovery briefly indicated.

THE application of geology to problems arising in coal mining engineering and coal preparation has, in general, been somewhat remote, or the geological problems have involved such simple forms of geological knowledge that the attention of specialists in this field has not been commonly thought necessary. Furthermore, the geology of coal beds and coal mining has received but scant attention in our educational system, even in mining education; so that coal mining engineers have become somewhat conditioned against geological thinking. Geologists, on the other hand, have commonly failed to realize that there are geological aspects to coal mining and coal preparation problems. A further reason why geology receives so little consideration in coal mining undertakings in this country lies in the fact that our coal supplies are, or have been thought to be, almost unlimited, have been easily accessible, and very cheaply mined.

Conditions have been changing within recent years. Failures can be less easily endured. Great inroads have been made on our bonanza deposits of easily mined coal and the increasing cost of coal and the increasing difficulty of finding it in easily accessible and large quantities places greater and greater importance on accurate information and good technique in exploration. It seems probable, therefore, that the opinions and aid of geologists will be more commonly sought than in the past in the exploration for coal and in the development and operation of coal mines, and analyses of mining conditions, preparation, and utilization.

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It may be timely, therefore, to extend to the Coal Division the recognition that geology receives in the Institute, in the fields of hard-rock mining, of petroleum engineering, and of nonmetallic-, nonfuel-resources engineering, evidenced by joint sessions with the sister society of Economic Geologists. It is my personal feeling that at least occasionally, with profit to both engineers and geologists, joint sessions might be arranged between the two societies similar to those set up with other divisions of the Institute, thereby bringing the two groups closer together in the field of common interest. A considerable number of the geologists who are members of the Coal Division are also members of the Society of Economic Geologists and have taken part in the activities of its Coal Research Committee. Such members and doubtless others would profit by sessions of the Coal Division in which geological aspects of coal occurrence, characteristics, distribution, roof conditions, and other matters relating to coal geology are considered. Such joint sessions would likewise provide a much needed stimulus to interest in coal geology in the academic field.

The field of coal geology stretches across the boundary between the two societies. Fundamental research in one group will have its eventual effect upon thinking and activities in the other group. Both societies have a real interest and desire to promote such research. The Committee of Coal Research of the Society of Economic Geologists believes that an account of the deliberations and activities of the Committee will be of interest to the members of the Coal Division. We might even obtain moral support. The main objective of the Committee is to stimulate interest in the geology of coal and to formulate a program of study and fundamental research along geological lines.

Coal geology may be conveniently organized with respect to six varieties of study and research: (1) coal resources investigations; (2) coal mining geology; (3) natural history of coal, origin of coal beds and of coal; (4) geobotanical investigations, coal botany and paleobotany; (5) coal petrography; and (6) coal geochemistry. A thorough understanding of these various branches of coal geology is not to be expected of a single individual but specialists in the different fields will develop if research is adequately supported. The content of each of the six fields of knowledge will be briefly considered.

Coal Resources Studies: Delineation and description of the location, distribution, thickness, structural and mining conditions of the coal deposits of areas of the earth have long been regarded as the prime concern of coal geology. It is the duty of the coal geologist engaged in coal resources studies to determine the facts in regard to the coal resources. On the basis of the accumulated information the engineer, or any other person that is qualified to do so, can estimate the reserves available for economic recovery under whatever conditions prevail at the time. The estimate of resources change only as information provides better understanding of the conditions of coal occurrence, distribution, and minability. The estimate of reserves which will usually be less than the resources may change from year to year as conditions determining mining costs and other factors in the competitive picture change. Within the province of these activities information is assembled relative to folds, faults, floor, and roof conditions, and the local character of and the regional variation in the chemical character of coal.

Coal Mining Geology: This phase of coal geology is largely unexplored but will be one of greater and greater importance as it becomes more and more necessary to extend mining operations in the preferred coal beds into areas where mining conditions are less favorable and less well known. Studies need to be extended to include observational and experimental investigations of the geological factors determining strength of roof. Various other aspects of mining geology are involved where beds are much folded or faulted, where water or gas is troublesome, where methods of exploration of coal properties require geophysical methods, and in the development of new properties, and in its interpretation of drill cores.

Natural History of Coal: Geological activities are also appropriately extended to include studies concerning the natural history of coal beds and of coal itself. What are the conditions under which plant debris accumulated? What portions of the plants withstood longest the geological vicissitudes of coalification? What parts were most readily "homogenized" to use a term and idea employed by Professor Wilhelm Petrascheck, of Leoben, Austria? These are questions which concern particularly those interested in the low-rank coals. Much European literature exists on the transformation of vegetable matter into coals of low-rank because such knowledge has been found useful in determining the utilization of such coals. It seems inevitable with the increasing relative importance of low-rank coals in this country that problems of initial coalification and early metamorphism will assume increasing importance. Undoubtedly solution of such problems will lead to a better understanding of the constitution of the higher rank coals.

Geobotanical Investigations: The use of plants and plant parts in fossil form in coal-bed correlation and the stratigraphic classification of coal measures has long been an established practice. In the past the evidence has consisted very largely of compressed fossil leaves, stems, and seeds found embedded in the shales and sandstones associated with the coal beds. Much information concerning the nature of the fossil plants has been obtained through study of coal balls in which plant material is found in a sort of petrified condition with the cellular parts only moderately compressed or entirely uncompressed. Within very recent years it has been demonstrated that the coal beds themselves provide fossils in the form of spores and pollen that in the case of Illinois coal beds, at least, can be used as type fossils for identifying coal beds. Possibly other fossils useful for identification may also be found in the coal. Such knowledge is of immediate practical importance where the bed being mined is crossed with faults and other beds of similar thickness are present above or below the bed being mined, or where explorations involve two or more closely spaced beds of similar thickness. Such geobotanical studies reveal the nature of the coal-producing flora and its variability. Studies that

have been made[1] point to the variable effect of the coalification process upon the organic material so that it may be found that rank is not determinable by the same criteria for all kinds of coal. In connection with the botanical studies important contributions have been made and will continue to be made in regard to the plant material in coal and the morphological characteristics of plants obtained in a fossilized condition in coal balls.

[1] James M. Schopf: Variable Coalification: The Processes Involved in Coal Formation. *Econ. Geol.* (1948) **43**, (3) 207-225.

Coal Petrography: Coal petrography or petrology is one of the later developments in the field of coal geology based very largely on the distinctions placed upon the different banded ingredients of coal by Dr. Marie Stopes in 1919.[2] The technique concerns the composition of coal in a manner analogous to the procedures of rock petrology. It is based primarily upon the recognition of the presence of four varieties of banded ingredients visible with the naked eye: vitrain, clarain, durain, and fusain. Coal petrography and coal botany overlap because it is possible to describe the composition of coal not only in terms of the megascopically-different banded ingredients but also on the basis of the microscopically-different components representing plant parts or secretions which the writer has called phyterals. This overlapping has resulted in a good deal of difficulty in terminology and description. Nevertheless, terms have been fairly well defined and the informed technician can shift from the petrographic to the botanical or phyteral terminology without much confusion.

[2] Marie C. Stopes. On the Four Visible Ingredients of Banded Bituminous Coal. *Proc. Roy. Soc.* (London) (1919) **90**, 470-487.

In describing a coal bed petrographically it is the usual procedure to note the distribution and proportion of the banded ingredients megascopically visible in a freshly broken, or cut and polished, column of the coal bed. Such studies give an idea of the composition of the bed petrographically in terms of vitrain, clarain, durain (splint), and fusain. The effect of variations in the proportion of the ingredients upon preparation, and eventual combustion, has been somewhat explored. Fusain is now recognized as the most important component of dust-forming material derived from the handling of mid-western coals. Vitrain in these coals undoubtedly has the greatest capacity for swelling and agglutinating, whereas durain (particularly splint durain) is noteworthy for its free-burning characteristics and lack of amenability to the process of hydrogenation. In fact, the capacity of a coal for hydrogenation seems to be largely determined by the amount of inert material with high carbon or low-volatile content, such as fusain and durain.

When a technique for the rapid analysis of broken coal in terms of the banded ingredients is developed, using unskilled or semiskilled labor, this method of analysis will doubtless have wider use if its practical value is also demonstrated. Progress in these directions is being made, but it must be admitted that with few exceptions, demand for such information has not developed. As noted above the fusain and the durain (splint) contents of coal have been regarded by some operators and some types of coal users as important and steps are taken to determine the amounts of these ingredients in some coals. Petrographic analyses made by the Illinois State Geological Survey of different sizes of broken coal shows quite clearly that ingredient composition differs from size to size and differs from the average composition of the bed. It is evident, therefore, that the mining process alone or the mining and preparations processes together are selective with respect to the banded ingredients. It is obvious, therefore, that the amount of the various ingredients present in the coal bed will determine the proportion of the various sizes produced. In some mining establishments this might be a consideration of importance. In general it is probably true that ingredient composition is less important in the higher than in the lower rank coals. Only exploratory studies have been made along this line, but they seem to indicate the importance of knowing something about the effect of mining and preparation processes upon the petrographic characteristics of the various sizes of the coal produced if there is a complete understanding of these processes. This is particularly the case with respect to the coals of medium and low rank. Up to the present there has been very little investigation of the possible effect of petrographic variation upon utilization except for special uses like briquetting and hydrogenation. Certain coal types resulting from petrographic variation, for example, splint, semisplint and cannel coals, have long been recognized in the coal trade and are used frequently for special commercial applications. Two benches of a single coal seam sometimes are mined separately to take advantage of this. Also various sizes from the same mine may go to different uses because of considerable segregation of petrographic contituents according to size.

Petrographic analysis may eventually, however, discover the explanation for differences in the day-to-day variations in coal that has received the same preparation or for the differences between coals produced by the same methods from different mines. In some cases the cause of variations may lie deeper than "ingredient consist," in which case recourse to microscopic examination may reveal botanical or phyteral differences in composition of significant importance. This is obviously an appropriate field of investigation where concern is with the fundamental physical makeup of coal and will probably be of greater and greater importance as more and more low-rank coals enter the market. It is one which might be appropriately explored in academic laboratories, but as far as I am aware is not being so explored. Work of this kind is, I believe, confined to two laboratories in this country, but a third is in the process of organization here in Columbus by the U.S. Geological Survey and is to be placed under the direction of Dr. James M. Schopf.

Coal Geochemistry: There is a natural transition from coal petrography to coal geochemistry. Coal petrography reveals a variety of coal components more or less reminiscent of various organic entities initially present in the plant debris from which the coal was formed. As coal is produced from various organic substances by geological processes, it is reasonable to inquire concerning the nature of the geologic processes involved and how they acted upon the raw material to produce the results ob-

tained. The process of metamorphism involves the application of pressure, more or less heat, one or both applied through a period of time; but it also involves the intimate events that transpire in the organic molecules subjected to the vicissitudes of the geological environment. The problem is that of determining the specifications of the environmental conditions and also of determining the reaction of the organic substances to these conditions. One of the natural consequences of a familiarity with the botanical and petrographic heterogeneity of the middle and low-rank coals is inquiry in regard to the part played by geological processes in producing, perpetuating, or eliminating such heterogeneity.

The failure to carry on such an investigation lies wholly with the geologists. The chemists cannot justly be taken to task for failure to conduct geological research. However, no such research can be carried on without the cooperation of chemists and particularly of chemists who have an understanding of and interest in geological phenomena and principles.

It is certain that for the indefinite future coal will be investigated predominantly by the standard procedures of the commercial types of analysis, by the application of various, carefully standardized tests for determining directly the adaptability of coal to various uses, by carefully developed procedures of solvent analysis, and by the conventional methods of organic analysis. A vast amount of extremely useful information will be added to that already existing. This information is of much value to geologists, for it is significant of geological phenomena in spite of its predominant empirical character. Because rank differences are fundamental with respect to coal, items of fact of a wide variety in regard to the properties of coal, even though empirically determined, can be fitted into the picture, and, in turn, be used empirically for description and classification purposes.

In the matter of coal chemistry, coal geologists have been short on a number of counts but I wish to point out two particularly: First, they have not succeeded in establishing the importance of the initial variability of coal-forming material to the extent of demonstrating its significance even in the middle and low-rank coals, where, it is probable such demonstration would be most readily possible. Second, they have not produced tangible criteria for evaluating the geological elements of the environment effecting metamorphism so that they can be used by the physical and structural chemist as a basis for theoretical consideration of their effectiveness in bringing about fundamental molecular changes in the components of coal. It is possible that these faults lie largely in the fact that American geologists interested in coal started at the wrong end of the coal series, where petrographic differences are of less definite significance. Whereas, if the start had been made in the low-rank coals, coal petrography and coal botany could have readily demonstrated the importance of coal heterogeneity and the definite relationship of component variability to variations in the coal. The changes in these components resulting from coalification could then have been followed through to the higher rank coals with the gradual tapering off of the influence of petrographic variability. The nature and importance of petrographic variations in coal would then probably have been better and more realistically understood.

The nature of coal geology has been described in brief outline. The various fields of study and research in many ways touch closely and not uncommonly considerably overlap the fields of coal-mining engineering and coal preparation and utilization. Research in those aspects of coal geology that impinge closely upon the practical fields of engineering in its various phases, such as the delineation and description of coal resources, description of various geological conditions affecting ease and cost of mining, studies in roof geology, and so on, has been fairly well supported particularly by Federal and state agencies. Even the somewhat more academic fields of coal botany and coal petrography occasionally contribute information of practical significance, particularly in the identification of beds and the interpretation of structural irregularities and in some phases of coal preparation and utilization. There is considerable activity in some of these fields and some activity in all, although not nearly as much as might be expected in view of the fact that, in general, colleges and universities, even those located in coal mining regions, give little or no attention to coal geology, particularly to research in this field.

The outstanding neglected field is that of coal geochemistry in spite of its obvious importance to an understanding of the nature of coal. It is primarily for the encouragement of research in this particular field that the Committee on Coal Research of our sister Society of Economic Geologists has declared itself. It appeared to this committee that the outstanding need in the field of coal geology and in the study of coal in general was fundamental research into the constitution of coal in which geologists and chemists would join as a geochemical project. It seemed to the Coal Research Committee, several of whom are members of the Institute, meeting in Pittsburgh in March 1949, in view of the fact that the interests of the two organizations overlap fairly widely, that it would be appropriate to make a statement to the Coal Division of the Institute in regard to the activities of the Coal Research Committee of the sister society, particularly with respect to its proposal for fundamental research. We are asking for no action on the part of the Coal Division. Should any of you be personally interested in the proposal and care to hear more about it, information will be supplied upon request. If you would like to be represented on a coal research committee that is endeavoring to build up general interest in coal geology, you are welcome to join our group, as there are no prerequisites aside from such interest.

I might say that the activities of the Committee extend somewhat beyond that of suggesting and promoting research of a fundamental sort, in that we are endeavoring to obtain wider recognition of coal geology in the programs of scientific and technical societies, and to encourage the inauguration of courses in coal geology in the colleges. To these ends a symposium on low-rank coals of the west was organized to be held at the time of the SEG and GSA joint meetings at El Paso in November 1949.

Ground Water in California

by J. F. Poland

Annual pumpage of ground water from the alluvial valleys of California now is about 10 million acre-feet. This heavy pumpage has created problems of overdraft and ocean-water encroachment in many valleys. Steps are being taken to protect and increase the available supply but much additional basic study is needed.

Location of Basins and Geologic Features of Occurrence: The major ground-water resources of California occur and are stored in the many large alluvium-filled valleys of the state. The deposits of Quaternary age which hold the ground water commonly are called valley fill. Meinzer[1] has discussed the characteristic features of this fill and showed that for western United States its distribution is concentrated chiefly in western Utah, Nevada, central Oregon, California, southern Arizona, and central New Mexico (see fig. 1). For California the principal ground-water basins are shown on fig. 2. Except for the Mojave Desert and the "Basin and Range" Valleys (fig. 2, Nos. 23 and 23A), which are not shown individually, the basins outlined on fig. 2 contain all the large reservoirs in valley fill within the state.

The Sacramento and San Joaquin Valleys (fig. 2) together comprise the Great Central Valley of California, which has an area of about 18,000 square miles. The Sacramento Valley contains about 5000 square miles of valley lands, approximately equal to the area of Connecticut; the San Joaquin Valley embraces about 13,000 square miles, equivalent to the areas of Massachusetts and Connecticut combined. Both the area and the ground-water use in the Central Valley is far greater than for all the other shaded alluvial basins of fig. 2 added together. Except for the Sacramento Valley, all the major ground-water basins of the state are south of Sacramento.

The valley fill in these ground-water basins has been transported by streams and most of it has been laid down in alluvial fans, producing the typical alluvial-fan structure of aquifers radiating as tongues from a prismatic pile of coarse, permeable debris at the canyon mouths; these aquifers are encased down slope by poorly sorted materials consisting of a clayey matrix with imbedded pebbles and boulders, interbedded with flood-plain silt. In the deeper inland structural valleys, the alluvial deposits interfinger at depth in the valley bottoms with horizontally bedded lacustrine sand, silt, and clay. The San Joaquin Valley is a good example. In the coastal valleys exposed to the sea, such as the Los Angeles coastal plain, the alluvial fans pass coastward into lagoonal and thence to shallow marine deposits. In all these basins, the gravel is coarsest and most permeable near the apexes of the fans, but tongues or lenses of clean gravel and sand may extend for many miles down the fan. Thus, these water-bearing structures are in a position to absorb stream flow near the canyon mouths and to transmit it through the buried radiating conduits.

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Especially in the valleys of southern California, the structure is complicated by faults which usually extend upward through the deposits of Pleistocene age. Where these faults cut impermeable confining beds, they have initially afforded passage for upward escape of artesian waters. The ascending waters undergo pressure decrease and, because the ground waters of California are characteristically of the calcium-bicarbonate type, carbon dioxide has escaped, causing precipitation of calcium carbonate and other salts together with silica. In this manner ground-water dams have developed even in wholly unconsolidated and highly permeable deposits. Gouge is believed to have been a negligible factor in the formation of these barrier features. In many of the basins of southern California these ground-water dams are primary factors in controlling the quantity of recharge and the position of the water level (and hence the pumping lift). For basins in the process of full development, these may be of primary consideration in determining the most effective utilization.

Fig. 1—Map of western part of United States showing principal areas underlain by Quaternary valley fill. (After Meinzer.)

Quantity Pumped and Uses: Ground-water development in California began in the Los Angeles area a few years after the end of the Civil War, about 1870. By 1900 approximately 10,000 wells had been put down within the four major valleys that comprise the South Coastal Basin.[3] According to Simpson[5] irrigation north of the Tehachapi Mountains did not become significant until after 1900. By the first World War, in the central as well as the southern part of the state, most of the surface water available from the unregulated streams for irrigation in the dry summer months had been diverted for use. Consequently, most of the development since then has been from the ground-water basins. This has been spurred by the great improvement in turbine pumps and the wide availability of electric power, as well as by the expanding population and the increasing demand for agricultural products.

The total amount of ground water pumped yearly from California's underground basins is not known accurately. However, the pumpage from all the basins outside the Central Valley is estimated to be approximately 2 million acre-feet (1 acre-foot = 325,851 gal). For the San Joaquin Valley, the combined gross pumpage of ground water from about 35,000 wells south of the Merced River for the one-year period from April 1, 1947, to April 1, 1948, has been estimated by the California Division of Water Resources from electrical energy consumed as close to 6 million acre-feet.[4] Of this, about one million acre-feet is pumped from the deep wells on the west side of the valley. Pumpage in the part of the San Joaquin Valley north of the Merced River and in the Sacramento Valley together is believed to be nearly two million acre-feet. Thus, the total gross pumpage in the state in the year 1947 to 1948 apparently was between 9,000,000 and 10,000,000 acre-feet. Of this total at least three-quarters was pumped from wells in the Central Valley. These figures suggest that the use of ground water in the state now is roughly equal to that of surface water, including Colorado River water. The pumpage is concentrated almost wholly in the months of little or no rainfall, from April to October. If uniformly distributed throughout the year, the discharge from all well pumps would be equal to about 13,000 second-feet, or about 8.5 billion gallons a day.

A comparison of current ground-water pumpage in California with the flow of main-stem streams and the storage capacity of large surface reservoirs is of use in visualizing its magnitude of draft. For example, the average gauged runoff of the Sacramento River at Red Bluff for the period 1896 to 1939 (water years) is 8,111,000 acre-feet a year.[5] At the other end of the state, the flow of the Colorado River is of critical interest. Morris[6] states

that for the past 50 years the average flow of the Colorado River at the Mexican border has approximated 17,720,000 acre-feet, under assumed virgin conditions. Thus, the estimated current ground-water draft in California is 10 to 20 pct higher than the 44-year average yearly discharge of the Sacramento River at Red Bluff and slightly more than half as great as the 50-year average runoff for the Colorado River.

Comparison with the capacity of surface-water reservoirs affords an instantaneous and simpler picture. For example, the current annual pumpage of ground water is about twice as great as the joint storage capacity of the reservoirs formed by Shasta and Friant Dams.

For the total gross pumpage of ground water, the distribution by major ground-water basins is approximately as follows: Central Valley, at least 75 pct; South Coastal Basin, 8 pct; Salinas Valley, 4 pct; Santa Clara County basins, 3 pct; Ventura County basins, 2 pct; Santa Barbara County basins, 1.5 pct; Antelope Valley and Coachella Valley, each about 1.3 pct; and miscellaneous small basins 1 to 2 pct.

The principal uses of ground water in California are threefold: in order of importance, they are for irrigation, for domestic supply, and for industry. For the year 1945 Guyton[7] has estimated that in California the draft in million gallons a day for industrial use was about 300 and that for municipal and rural use about 350, equivalent respectively to about 330,000 and 390,000 acre-feet a year. Although the rapid influx of people to California must have increased domestic use of ground water substantially since 1945, it is believed that combined

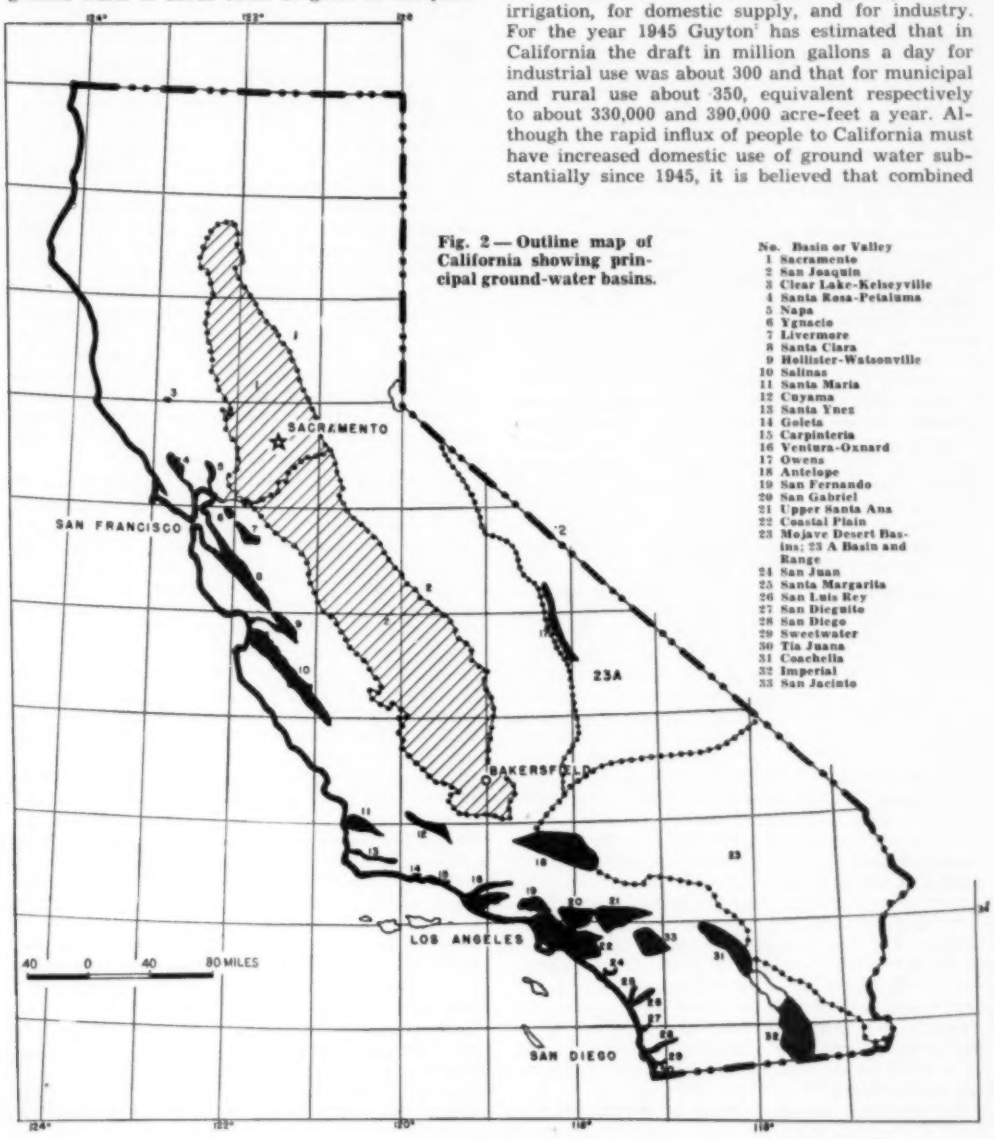

Fig. 2 — Outline map of California showing principal ground-water basins.

No.	Basin or Valley
1	Sacramento
2	San Joaquin
3	Clear Lake-Kelseyville
4	Santa Rosa-Petaluma
5	Napa
6	Ygnacio
7	Livermore
8	Santa Clara
9	Hollister-Watsonville
10	Salinas
11	Santa Maria
12	Cuyama
13	Santa Ynez
14	Goleta
15	Carpinteria
16	Ventura-Oxnard
17	Owens
18	Antelope
19	San Fernando
20	San Gabriel
21	Upper Santa Ana
22	Coastal Plain
23	Mojave Desert Basins; 23 A Basin and Range
24	San Juan
25	Santa Margarita
26	San Luis Rey
27	San Dieguito
28	San Diego
29	Sweetwater
30	Tia Juana
31	Coachella
32	Imperial
33	San Jacinto

domestic and industrial use still is somewhat less than one million acre-feet a year. Evidently at least 90 pct of the ground water pumped is used for irrigation.

The pumpage of ground water in all parts of the United States has increased greatly in the past decade. In 1945, according to Guyton,[7] the average daily use of ground water was about 20 billion gallons, equivalent to 22 million acre-feet for the year, and about twice the draft in 1935.[8] Doubtless the country-wide draft has increased substantially since 1945, but even if it has risen as much as a third, the draft in California is now about one-third of the total. Arizona is second in rank with pumpage of ground water in 1947 estimated by Turner[9] as on the order of 2.6 million acre-feet.

Problems Created by Intensive Development: The long-continued and increasing demands on the ground-water supplies of the state have created serious problems in many of the ground-water basins. In some of the basins, water levels have been drawn down many scores and even hundreds of feet; in a few coastal basins the drawdown has pulled water levels many feet below sea level. The principal problems created by this great lowering of water levels and the application of large quantities of ground water and surface water to irrigation of the agricultural lands are those of (1) overdraft, (2) salt-water encroachment, (3) gradual changes in the chemical and physical character of the soils, and (4) long-term changes in the chemical character of ground waters. Only the first two are considered here. Protection of the ground-water supplies from pollution by industrial wastes has become a very important problem in recent years, owing to the great increase in industrial plants and somewhat indiscriminate disposal of wastes. That problem, which is now under investigation by several state agencies, has developed primarily because of rapid industrial expansion.

Overdraft: Prolonged and progressive drawdown of water level in a ground-water basin suggests an overdraft, that is, that the average net draft is exceeding the average replenishment. Such a condition is more noticeable if the depth to water is rapidly approaching the economic limit of pumping or if the drawdown is dewatering a substantial part of the saturated thickness of water-bearing beds.

In California, the problem of overdraft is complicated by the distribution of rainfall, which is the most basic factor in recharge. Not only is the rainfall concentrated almost wholly in the months from October to May, but also it varies greatly from year to year and has a rude cyclical distribution, with periods of many years in which the average rainfall is deficient and others in which it is above normal. For this reason both recharge and draft fluctuate not only from year to year but from period to period. For the most completely developed ground-water basins of the state, those in the Los Angeles area, problems of overdraft and the complications caused by this rude cyclical distribution of rainfall have been analyzed in an exhaustive report by Gleason.[10]

An apparent overdraft may be wiped out by great replenishment in a very few excessively wet years. For example, water levels in wells in the San Gabriel Basin (fig. 2) above Whittier Narrows were drawn down more than 100 ft during the two decades from 1916 to 1936, which was a period of deficient rainfall. Toward the end of this period it was considered by some that an overdraft existed. However, during the wet period from 1936 to 1945, the rainiest 10-year period on record for the Los Angeles area, the excessive runoff and ground-water recharge refilled the basin and restored water levels to those of 1916. The prolonged drawdown of the water table and the long-continued use of stored water prior to 1937 made available a great underground reservoir for retention of subsequent recharge. In this case, haphazard exploitation produced a classic example of the advantages and use of a ground-water basin for cyclic storage of water.

In at least a dozen areas in California a condition of real overdraft now exists. The largest is the San Joaquin Valley. Here the areas of overdraft on the east side are chiefly south of the Kings River recharge area, south of Hanford and Visalia. In this area the overdraft probably is within the range of half a million to a million acre-feet a year. On the west side of the valley, for the 150-mile strip from Mendota to the south end, development of ground water has taken place largely since 1935 and has been most rapid since the early forties. Here the pumpage is now on the order of a million acre-feet and probably more than half of the water is being mined. Compared to the discharge of Sierra streams to the east, the flow of the west side streams is very low and recharge is small.

Other large basins of the state which are now reported as or known to be overdrawn are the Santa Clara Valley fringing San Francisco Bay;[11] the lower east side of the Salinas Valley;[12] the Santa Maria Valley;[13] the Ventura-Oxnard area;[14] the Antelope Valley;[15] the Upper Santa Ana Valley and the main and West Basins of the coastal plain, which are in the South Coastal Basin.[16]

Salt-water Encroachment: *Ocean-water Intrusion:* Several of the major and at least a dozen of the minor ground-water basins of California border the coast and their seaward margins are in contact with or extend beneath the Pacific Ocean or its landward salients such as San Francisco Bay. Geologic and hydrologic evidence accumulated to date show that in these basins the seaward ends of the aquifers are in hydraulic contact with the ocean. Under natural conditions of seaward hydraulic gradient, fresh ground water was discharged to the sea or to its inland extensions. In places along the coast where heavy pumping has drawn down the water level below sea level, the hydraulic gradient has been reversed and ocean water has invaded the aquifers either directly or through defective wells.

In this way, salt water has invaded to some degree ground-water basins in the following areas: the Sacramento-San Joaquin delta between Antioch and Pittsburg; the Napa Valley; the Santa Clara Valley from Oakland on the east to San Mateo on the west;[16] the mouth of the Salinas Valley;[12] the Ventura area; the coastal plain of Los Angeles and

Orange Counties, both in the West Basin,[17] and in the main coastal basin;[18] and several of the small coastal basins of San Diego County,[19] especially the Tia Juana Valley.* The most serious invasion to date is that in the coastal plain of Los Angeles and Orange Counties. In the West Basin between Redondo Beach and Santa Monica ocean water is advancing inland along a 10-mile front and has progressed as much as two miles from the coast; in the main coastal basin ocean water has advanced inland as much as two miles in Santa Ana Gap of Orange County, and the potential threat to the Orange County supply is great.

* Ground-water supply of Tia Juana Valley in San Diego County is now under adjudication and problems of salt-water intrusion are being investigated by the California Division of Water Resources, under the direction of G. B. Gleason.

Interior Contamination: In several basins, saline contamination has developed by invasion of interior saline waters apart from the ocean. For example, in the vicinity of Marysville in the so-called peach bowl of the Sacramento Valley it is reported[20] that at places excessive drawdown of the water level in wells has brought up saline water from below. Also on the west side of the San Joaquin Valley in the vicinity of Vernalis and to the south, beds containing saline waters locally underlie the fresh-water aquifers. Wells which penetrated too great a thickness of water-bearing beds have had to be plugged so as to seal off the undesirable bottom waters. Locally in the Salinas Valley,[21] intensive drawdown of the pressure surface in confined aquifers has permitted saline shallow waters to move down and contaminate the fresh waters beneath, either through defective well casings or by direct leakage through the confining beds.

Methods for Increasing or Protecting the Available Supply: *Importation and Recharge for Cyclic Storage:* Overpumping of a ground-water basin and progressive lowering of the water level inevitably will develop if the average net long-term discharge, both natural and artificial, is greater than the average net recharge. There are only two possibilities for decreasing or eliminating an overdraft: (1) decrease the net discharge, or (2) increase the recharge.

In the early stages of development, general regional drawdown of the water level in a basin may substantially decrease natural discharge, both subsurface outflow and discharge by plants. After this has been accomplished, however, the only means of reducing net discharge still further is by decreasing pumping draft. This never will be a popular action and probably never would succeed unless supported by legal authority or enforced by economic necessity.

Fortunately, the recharge ordinarily can be increased substantially by conservation measures, by catching the runoff in detention reservoirs and spreading it on permeable deposits so that it will go into storage in the ground-water basins. In some basins, however, it is too costly to retain much of the flash runoff in surface retention dams, and in a few insufficient runoff is available. In such cases, importation of water is the only means for supplementing the ground-water supply.

In southern California importation of water from the Colorado River in quantities much larger than the gross ground-water draft of that area is taking care of many of the expanding needs for the present and immediate future, except for basins like the Antelope Valley, which has no immediate prospects for amelioration of its serious overdraft.

The water deficiency in the San Joaquin Valley, which existed as early as 1921, was one of the major incentives for the State Water Plan investigations which were begun in that year. The Central Valley Project has developed from these investigations. As summarized by Hyatt.[22]

> This plan included the creation of a large storage reservoir, on the Sacramento River at the head of the valley near Redding, which would regulate the flow of the river, provide flood control, and generate a large amount of hydroelectric power. Waters released from this reservoir during the summer would flow down the Sacramento River, eliminate irrigation deficiencies and assure navigation in that valley throughout the year, overcome the encroachment of salinity in the Sacramento-San Joaquin Delta, and provide sufficient additional water for diversion southward by means of pumps into the San Joaquin Valley. . . . A second reservoir was also planned on the San Joaquin River at the town of Friant, north of Fresno, with large canals leading north and south to supply the distressed areas whose problems were the main reason for the preparation of the plan.

It was concluded for the San Joaquin Valley that the flood runoff could not all be conserved economically in surface reservoirs and that irrigation requirements during years of deficient rainfall could be supplied only by taking ground water from storage.[23] As reported by Livingston,[24] it was pointed out[25] that an assured water supply for the southern half of the east side of the Valley requires the surplus runoff of wet years to be detained underground in cyclic storage for withdrawal and use in dry years. The volumes of ground-water storage required for use in dry periods is very large, as much as 16,000,000 to 18,000,000 acre-feet. It has been assumed that this water can be put underground in a few wet years by excess irrigation, assisted by percolation from stream channels, canals, and ditches.

Little is known as yet about the structure of the water-bearing deposits in the San Joaquin Valley and about the movement and circulation of ground water in the aquifers that lie within that structure. The Central Valley Project is now under construction by the Bureau of Reclamation. The success of that project as it affects the San Joaquin Valley depends fundamentally on the ability to put underground this cyclic storage at rates of as much as 3,000,000 acre-feet or more a year. So far investigators of the Bureau of Reclamation and other agencies have obtained only a part of the basic information on ground-water conditions and on the other factors affecting the practicability of artificial recharge in the large volumes contemplated.

In connection with the problems of overdraft and need for additional water in many of the ground-water basins of the state, a reappraisal of the overall water resources is now under way by the State Water Resources Board through the Division of

Water Resources. The primary purpose is to determine where surplus waters exist, how great they are, and how they can be utilized to best advantage in areas of deficient supply.

Restraint of Saline Encroachment: In ground-water basins where no general overdraft exists, drawdown of water levels along the coast may cause a relatively minor intrusion of ocean water that will not pass inland appreciably beyond the area where there is a landward gradient. Such apparently is the case in the Salinas Valley.[12] On the other hand, if a basin is pumped so heavily that there is a basin-wide depression of water levels below sea level, ocean-water intrusion can continue to march steadily across the basin so long as the water levels remain depressed. This is the situation in the West Basin of Los Angeles County.[17] In most ground-water basins, such a condition could be remedied most effectively in the long run by restoring water levels throughout the basin to a sufficient height that the salt water would be displaced seaward. The West Basin is anomalous, however, in that about half its current recharge is derived by underflow through the barrier faults of the Newport-Inglewood uplift, its inland boundary. The recharge by underflow continues only because there is a water-level differential across the faults ranging from 30 to 60 ft, with water levels on the inland side about at sea level and those in the West Basin 30 to 60 ft lower. Thus, if levels on the coastal side (within the West Basin) are raised to or above sea level, this recharge will cease. Therefore, to utilize the basin effectively, it is possible that artificial recharge could be employed to build a freshwater ridge near the coast and pumping could be concentrated along the inland edge so as to maintain the favorable water-level differentials and the recharge.

Restraint of ocean-water intrusion by artificial recharge and the development of coastal freshwater ridges ultimately may prove economically feasible for several ground-water basins along the coast. Geologic conditions will determine whether this can be accomplished by spreading water at the land surface for percolation from trenches, pits, or basins, or by the more expensive methods of recharge through wells. An engineering analysis based on the facts would determine whether such measures are economically feasible.

Artificial recharge has been practiced in California for nearly 50 years. Most of the operations, for obvious reasons, have been concentrated at the permeable intake areas at or near the heads of the alluvial fans. In the future, the steps which are taken to increase or protect the supplies of ground water in California doubtless will result in a great increase in the quantities of water placed in cyclic ground-water storage, and necessarily will require much research on methods and techniques of artificial recharge. As a prerequisite to effective progress, the geology and hydrology of the critical ground-water basins should be examined in great detail.

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Studies of the

Effect of Freight Rates on Marketing

Northwest Industrial Minerals

by Leslie C. Richards

The competitive position of producers of industrial minerals depends upon the delivered price of their product. Freight charges are a major factor in the sales to consumers. A comparison of freight rates on movements of limestone and silica rock in the Pacific Northwest shows a wide variance in mills per ton mile for like hauls, thereby favoring some localities over others.

Introduction: Sooner or later the potential miner of industrial minerals reaches the conclusion that of the various factors he must consider, the marketing of his product is the most important. The sooner he realizes this the better, as successful industrial mineral producers will confirm. Herein lies a difference between industrial mineral mining and metal mining. Often in the latter, technical problems of mining or concentrating are equal to or even surpass those of marketing. This is not meant to imply that there are no problems in mining and beneficiation of industrial minerals, but these usually are outweighed by the problem of marketing.

Recently, an official of a nationally known corporation in the electro-metallurgical industry made an industrial mineral survey in the Pacific Northwest. In the course of his investigation he interviewed producers and potential producers of the various industrial minerals scheduled for use in the plant now being constructed in this area. He stated that he was interested in only three factors: namely, (1) Can your product meet our specifications? (2) Are you able to supply our tonnage requirements? and (3) What will be the delivered price? If there are several sources of supply of sufficient grade and tonnage, as is often the case in the Northwest, then the third factor becomes the important and deciding one.

While industrial minerals are usually sold f.o.b. cars with the freight being paid by the buyer, it is the delivered and not the f.o.b. price that determines the competitive position of the producer and his industrial mineral product. Unfortunately for both the consumers and producers of industrial minerals in the Northwest, the sources are generally a considerable distance from the markets. This necessitates long rail hauls, for which transportation charges make up the largest share of the delivered cost and the delivered price of the product.

Is it any wonder then, that rail rates play such an important part in the marketing of industrial minerals? In order to sell his product, the producer must be able to deliver to the market at a competitive price. It must be remembered that his largest cost factor is rail transportation, and that he is limited to markets that he can reach competitively.

Since the cost of rail transportation is so important, let us compare various freight rates in the Northwest and attempt to determine their influence on marketing. A study of all rates on all industrial minerals transported in this area, is beyond the scope of this paper. For the purposes of comparison, let us consider limestone (broken, crushed, or ground), and silica rock. These are widely used industrial minerals, and are carried by several different rail routes from the sources of supply to the centers of consumption in the Pacific Northwest.

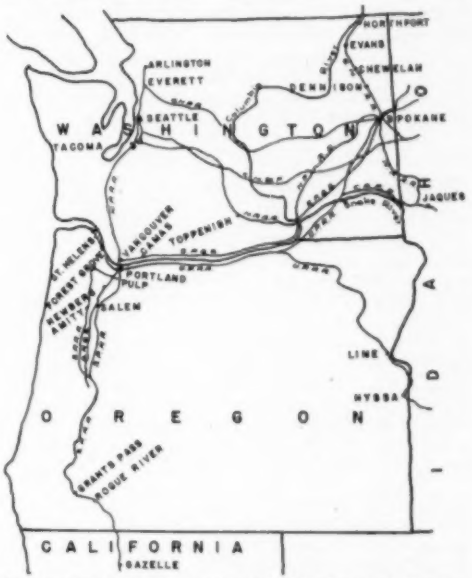

Fig. 1—Railroads that serve Washington and Oregon.

Markets and Sources of Supply: Limestone has a number of uses in western Oregon and Washington. Among them are agricultural lime for the soils of the Willamette Valley and southwestern Washington; paper rock for the mills in such cities as West Linn, Oregon City, Newberg, St. Helens, and Salem, Oregon, and Camas and Everett, Wash.; chemical and metallurgical-grade limestone for the metallurgical plants centering around Portland, and the Tacoma-Seattle areas; and sugar rock for the sugar refineries at Toppenish, Wash., and Nyssa, Oregon. Since limestone for cement manufacture is generally processed near the quarry, transportation

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Table I. Railroad Rates on Limestone, Pacific Northwest*

To	From	Use	Miles	Tariff	Minimum Carload, Lb	Railroads	Mills Per Ton Mile
Amity, Ore.	Gazelle, Calif.	Ag. lime	344	$3.45	100,000	SP	10.0
Amity, Ore.	Grants Pass, Ore.	Ag. lime	231	2.90	80,000	SP	12.5
Salem, Ore.	Gazelle, Calif.	Ag. lime and paper rock	357	3.45	100,000	SP	9.7
Salem, Ore.	Grants Pass, Ore.	Ag. lime and paper rock	244	2.90	100,000	SP	11.9
Salem, Ore.	Jaques, Ida.	Ag. lime and paper rock	426	4.50	100,000	Camas P. RR, UP, SP	10.6
Salem, Ore.	Northport, Wash.	Ag. lime and paper rock	557	4.45	100,000	GN, SPS, SP	8.0
Salem, Ore.	Evans, Wash.	Ag. lime and paper rock	534	5.05	100,000	GN, SPS, SP	9.4
Salem, Ore.	Arlington, Wash.	Ag. lime and paper rock	300	4.13	60,000	NP, SP	13.8
Oswego, Ore.	Lime, Ore.	Ag. lime	394	2.87	5 cars	UP, SP	7.3
Forest Grove, Ore.	Gazelle, Calif.	Ag. lime	448	3.85	100,000	SP	8.6
Forest Grove, Ore.	Grants Pass, Ore.	Ag. lime	335	2.95	100,000	SP	8.8
Forest Grove, Ore.	Jaques, Ida.	Ag. lime	411	5.40	100,000	Camas P. RR, SP	13.1
Vancouver, Wash.	Gazelle, Calif.	Ag. lime and chemical grade	420	5.70	100,000	SP, UP	13.6
Vancouver, Wash.	Grants Pass, Ore.	Ag. lime and chemical grade	307	3.50	100,000	SP, UP	11.4
Vancouver, Wash.	Lime, Ore.	Ag. lime and chemical grade	385	3.25	80,000	UP	8.4
Vancouver, Wash.	Jaques, Ida.	Ag. lime and chemical grade	373	3.25	80,000	Camas P., UP	8.7
Vancouver, Wash.	Northport, Wash.	Ag. lime and chemical grade	496	3.85	80,000	GN, SPS	7.8
Vancouver, Wash.	Evans, Wash.	Ag. lime and chemical grade	471	3.85	80,000	GN, SPS	8.2
Vancouver, Wash.	Arlington, Wash.	Ag. lime and chemical grade	237	2.25	60,000	NP	9.5
Vancouver, Wash.	Orcas Island	Ag. lime and chemical grade		3.85	60,000	Barge and NP	
Pulp, Ore.	Gazelle, Calif.	Paper rock	392	3.85	100,000	SP	9.8
Pulp, Ore.	Grants Pass, Ore.	Paper rock	279	2.90	80,000	SP	10.4
Pulp, Ore.	Jaques, Ida.	Paper rock	391	3.85	100,000	Camas P. RR, UP, SP	9.8
Pulp, Ore.	Northport, Wash.	Paper rock	522	4.45	100,000	GN, SPS, SP	8.5
Pulp, Ore.	Evans, Wash.	Paper rock	499	4.45	100,000	GN, SPS, SP	8.9
Pulp, Ore.	Arlington, Wash.	Paper rock	265	3.58	60,000	NP, SP	13.5
Newberg, Ore.	Gazelle, Calif.	Paper rock	380	3.85	100,000	SP	10.1
Newberg, Ore.	Grants Pass, Ore.	Paper rock	267	2.90	80,000	SP	10.9
Newberg, Ore.	Jaques, Ida.	Paper rock	393	4.05	100,000	Camas P. RR, UP, SP	10.3
Newberg, Ore.	Northport, Wash.	Paper rock	524	5.80	80,000	GN, SPS, SP	11.1
Newberg, Ore.	Evans, Wash.	Paper rock	501	5.80	80,000	GN, SPS, SP	11.6
Newberg, Ore.	Arlington, Wash.	Paper rock	267	3.69	60,000	NP, SP	13.8
St. Helens, Ore.	Grants Pass, Ore.	Paper rock	324	4.50	100,000	SP, SPS	13.9
St. Helens, Ore.	Evans, Wash.	Paper rock	509	3.85	80,000	GN, SPS	7.6
Camas, Wash.	Arlington, Wash.	Paper rock	252	2.55	100,000	NP, SPS	10.0
Camas, Wash.	Gazelle, Calif.	Paper rock	435	5.70	100,000	SP, SPS	13.1
Camas, Wash.	Grants Pass, Ore.	Paper rock	322	3.50	100,000	SP, SPS	10.9
Camas, Wash.	Jaques, Ida.	Paper rock	398	3.25	80,000	Camas P. RR, NP, SPS	8.2
Camas, Wash.	Northport, Wash.	Paper rock	479	3.05	100,000	GN, SPS	6.4
Camas, Wash.	Evans, Wash.	Paper rock	456	3.15	100,000	GN, SPS	6.9
Camas, Wash.	Orcas Island	Paper rock		3.85	60,000	NP, SPS	
Portland, Ore.	Gazelle, Calif.	Chem. and met. grade	410	3.85	100,000	SP	9.4
Portland, Ore.	Grants Pass, Ore.	Chem. and met. grade	297	2.95	100,000	SP	10.0
Portland, Ore.	Lime, Ore.	Chem. and met. grade	385	3.25	80,000	UP	8.4
Portland, Ore.	Lime, Ore.	Chem. and met. grade	385	2.87	5 cars	UP	7.4
Portland, Ore.	Jaques, Ida.	Chem. and met. grade	373	3.25	80,000	Camas P. RR, UP	8.7
Portland, Ore.	Northport, Wash.	Chem. and met. grade	504	3.85	80,000	GN, SPS	7.6
Portland, Ore.	Evans, Wash.	Chem. and met. grade	481	3.85	80,000	GN, SPS	8.0
Portland, Ore.	Arlington, Wash.	Chem. and met. grade	247	2.25	60,000	NP	9.1
Portland, Ore.	Orcas Island	Chem. and met. grade		4.05	60,000	Barge and NP	
Seattle, Wash.	Jaques, Ida.	Chem. and met. grade	423	3.25	80,000	Camas P. RR, NP	7.7
Seattle, Wash.	Northport, Wash.	Chem. and met. grade	454	3.85	80,000	GN	8.5
Seattle, Wash.	Evans, Wash.	Chem. and met. grade	431	3.85	80,000	GN	9.0
Seattle, Wash.	Arlington, Wash.	Chem. and met. grade	61	1.40	100,000	NP	23.0
Seattle, Wash.	Kendall, Wash.	Chem. and met. grade	140	2.35	100,000	CMStP, NP	16.8
Seattle, Wash.	Orcas Island	Chem. and met. grade		2.20	100,000	Barge	
Everett, Wash.	Jaques, Ida.	Paper rock	456	3.85	80,000	Camas P. RR, NP, GN	8.4
Everett, Wash.	Arlington, Wash.	Paper rock	30	1.40	100,000	NP	46.8
Everett, Wash.	Kendall, Wash.	Paper rock	110	2.35	100,000	CMStP, NP	21.3
Toppenish, Wash.	Jaques, Ida.	Sugar rock	243	2.45	80,000	Camas P. RR, NP	10.0
Toppenish, Wash.	Northport, Wash.	Sugar rock	340	3.85	80,000	GN, NP	11.3
Toppenish, Wash.	Evans, Wash.	Sugar rock	317	4.45	60,000	GN, NP	14.0
Nyssa, Ore.	Lime, Ore.	Sugar rock	155	1.28	100,000	UP	23.3

* Rates include Ex-162 and 166 increases but not Ex-168.
Tariff references: Item 7455 SFTP 1-S; Sp Co. 233-H; SP Co 235-H; Bohons 5-U, I-1662; SPS 820; Bohons 2-K; Bohons 65-N; Bohons 1-N; Puget Sound Freight Lines 23-C.

is not a major problem. However, limestone for the other uses mentioned either is or has been shipped from the following localities: Gazelle, Calif., south of Yreka; Grants Pass, Oregon; Lime, Oregon, near Huntington; Evans and Northport, Wash., north of Spokane; Arlington, Wash., north of Everett; Jaques, Idaho, east of Lewiston; and the San Juan and Orcas Islands in Puget Sound.

Silica rock for the Portland-Vancouver electrometallurgical industries is shipped from Denison, Wash., north of Spokane, and Rogue River, Oregon, near Grants Pass.

Briefly then, the sources of supply of limestone and silica rock, with few exceptions, are in southern and eastern Oregon, northern California, Washington, and Idaho, while the principal markets for these sources are in northwestern Oregon and western Washington. The rail hauls involved range from 200 to 500 miles.

Railroads Serving Area: The states of Oregon and Washington are served by the lines of five major railroads, and by several local roads that are, for the most part, subsidiaries of, or at least controlled by, the major roads (fig. 1). The five major railroads are the Southern Pacific, Union Pacific, Northern Pacific, Great Northern, and the Milwaukee

(Chicago, Milwaukee, St. Paul and Pacific). The Southern Pacific, serving western Oregon, enters the state from California and terminates at Portland. The main line of the Union Pacific crosses the northern part of Oregon from the east to Portland and then turns north to Tacoma and Seattle. A branch of this railroad also reaches Spokane from eastern Oregon. The Northern Pacific, Great Northern, and Milwaukee roads enter the Puget Sound industrial area from eastern Washington. The first two continue south to the Portland-Vancouver area.

Local lines of interest are the Spokane, Portland and Seattle, the Camas Prairie, and the Oregon Electric, now a division of the S.P. and S. The S.P. and S. is owned jointly by the Northern Pacific and the Great Northern. It follows the north bank of the Columbia River and is a direct route from Spokane to Portland. The Camas Prairie serves the central Idaho region from Riparia, Wash., to Grangeville, Idaho. It is controlled jointly by the Union Pacific and the Northern Pacific, and connects with the lines of both. The Oregon Electric is an entry into the Willamette Valley for the Northern Pacific and the Great Northern. As Portland is the terminus for the major rail lines, joint line-haul and switching charges are such that, in general, shippers from the east and north cannot compete in markets south of Portland, and shippers from the south cannot compete north of Portland.

Freight Rate Structures: The average individual has little or no knowledge of the mechanics of freight rates. This is not surprising, as freight rate structures, particularly commodity rates, of the railroads are so complex as virtually to defy description except in terms of particular commodities. Freight moves under either a class or a commodity rate. Where there are no published commodity rates, limestone and silica rock move under a class E rate, the lowest of the class rates. The great bulk of traffic moves under, and most of the freight revenue is produced by, commodity rates. These are specific rates quoted directly, rather than through the medium of classification, on designated services, that for one reason or another require special rates. A commodity rate is usually lower than the class rate it displaces and applies to large shipments moving long distances, to low-grade articles taking short hauls, and to especially competitive traffic. The Interstate Commerce Commission, Class Rate Investigation, Docket No. 28300, exhibit No. 228 (1942) states: "Of the carloads of freight originating on September 23, 1942, on all railroads in the U.S., 85.2 percent moved under commodity rates. Commodity rates accounted for 77.6 percent of the revenue from carload traffic."

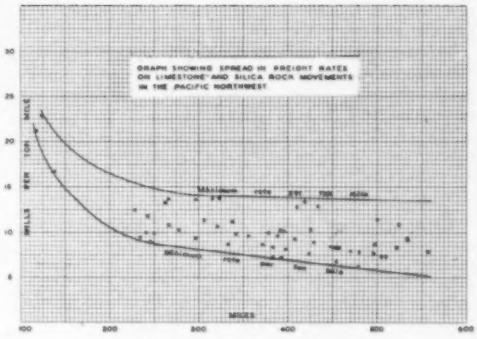

Fig. 2—Spread in freight rates.

Since we are interested in commodity rates and the way they affect shipments of limestone and silica rock in the Northwest, the question arises as to who sets these rates and on what basis. A shipper of limestone or silica rock, if he can show that there will be a sufficient movement of his commodity, can request a rate from the railroads involved to the point of destination. The railroads' freight rate-making agent in the Northwest, which is the North Pacific Coast Freight Bureau, then sets and publishes a rate under the supervision of the Interstate Commerce Commission. Some freight rate factors used are: (1) cost of service to the carrier, (2) value of service to the shipper, (3) value of the article, (4) nature of the article, (5) risk in handling the article, (6) distance of the haul, (7) bulk and weight of article, (8) whether special facilities or extra services are required, (9) volume of traffic and periods of movement, (10) rates on similar articles under similar circumstances and conditions, (11) competition between producing centers, (12) whether or not movement will be based upon rate, and (13) prospects of car being returned loaded or empty.

In practice, rates have not always been fixed at the best levels, nor is it likely that they ever will be exactly scientific. Rate making is an art and involves the exercise of judgment. Absolute standards are not available and between the upper and lower limits of reasonableness may lie a consider-

Table II. Railroad Rates on Silica Rock, Pacific Northwest[a]

To	From	Use	Miles	Tariff	Minimum Carload, Lb	Railroads	Mills Per Ton Mile
Portland, Ore.	Rogue River, Ore.	Met. grade	306	$2.90	100,000	SP	9.6
Portland, Ore.	Dennison, Wash.	Met. grade	401	3.15	100,000	GN, SPS	7.8
Portland, Ore.	Chewelah, Wash.	Met. grade	439	3.45	100,000	GN, SPS	7.8
Vancouver, Wash.	Rogue River, Ore.	Met. grade	316	4.23	60,000	SP, SPS	13.4
Vancouver, Wash.	Dennison, Wash.	Met. grade	391	3.15	100,000	GN, SPS	8.1
Vancouver, Wash.	Chewelah, Wash.	Met. grade	429	3.45	100,000	GN, SPS	8.1
Seattle, Wash.	Rogue River, Ore.	Met. grade	498	5.45	60,000	SP, GN	10.9
Seattle, Wash.	Dennison, Wash.	Met. grade	350	2.85	100,000	GN	8.2
Seattle, Wash.	Chewelah, Wash.	Met. grade	378	2.85	100,000	GN	7.6
Spokane, Wash.	Dennison, Wash.	Met. grade	20	1.25	100,000	GN	63.0
Spokane, Wash.	Chewelah, Wash.	Met. grade	58	1.25	100,000	GN	21.6

[a] Rates include Ex-162 and 166 increases but not Ex-168.
Tariff references: SP 233-H; Bohons 2-K; Bohons 5-U.

able zone. Rates are said to be governed by the three C's: competition, comparison, and compromise.

Former Commissioner Eastman of the Interstate Commerce Commission expressed his idea on the origin of rates when he said in 1925, "The railroad rate structure of the United States is the product of the efforts of freight-traffic managers seeking maximum revenue of their respective railroads, modified to some extent by public regulation." (100 I.C.C. 513611.)

Comparison of Rates: A study of local rail rates on limestone and silica rock shows a wide variance in mills per ton mile even for hauls of like distances. Fig. 2, with the charge per ton mile plotted against the distance hauled, clearly shows the spread in freight charges on limestone and silica rock movements in the Pacific Northwest. Most of the movements fall within the range of 250 to 500 miles. For these distances, the maximum charges per ton mile are roughly twice the minimum charges for the same distance.

These differences raise a question in the minds of laymen, particularly when other factors appear the same, such as one-line or joint-line hauls. Here note the following examples:

Commodity hauled: Limestone.

	Distance, Miles	Tariff	Mills Per Ton Mile	Railroad
Grants Pass to Amity, O.	231	$2.90	12.5	SP
Arlington to Vancouver, W.	237	2.25	9.5	NP
Grants Pass to Salem, O.	244	2.90	11.9	SP
Arlington, W., to Portland, O.	247	2.25	9.1	NP

In some cases joint hauls are lower than single hauls of like distances. If one were to believe that the various freight-making factors were used to determine the tariffs, the reverse should be true, but note the following examples:

Commodity hauled: Limestone.

	Distance, Miles	Tariff	Mills per Ton Mile	Railroad
Jaques, I., to Seattle W.	423	$3.25	7.7	Camas P. & NP
Northport to Camas, W.	479	3.05	6.4	SP & S & GN
Gazelle, C., to Portland, O.	410	3.85	9.4	SP
Lime to Portland, O.	385	3.25	8.4	UP

In some cases, rates on joint hauls to greater distances are the same as rates on a single haul for part of the distance, and in other cases the reverse is true, as can be seen from the following examples:

Commodity hauled: Limestone.

	Distance, Miles	Tariff	Mills Per Ton Mile	Railroad
Arlington to Vancouver, W.	237	$2.25	9.5	NP
Arlington to Camas, W.	252	2.55	10.0	NP & SP&S
San Juan Is. to Vancouver, W.		3.85		NP*
San Juan Is. to Camas, W.		3.85		NP & SP&S*

* By barge to Seattle.

Other examples of inconsistencies are:

(1) The comparison of hauls from Northport, Wash., and Jaques, Idaho. The tariff from Northport to Camas is $3.05, and to Portland $3.85; while the tariff from Jaques to Camas is $3.25 and to Portland it is also $3.25. Both hauls are handled by the SP&S railroad.

(2) The tariff from Northport, Wash., to Camas, Wash., is $3.05. That from Evans, Wash., to Camas is $3.15. Both loading points are on the same rail route. The only difference is that Evans is 23 miles closer to Camas.

(3) In the case of hauls from Denison and Chewelah, Wash., of silica rock the rates to Spokane are both $1.25. They are both $2.85 to Seattle, but to Portland, the tariff from Denison is $3.15, while that from Chewelah is $3.45.

Conclusion

Limestone and silica are low value commodities which are handled by railroads in large volume with standard equipment at minimum risk and expense. Both products are shipped in bulk in the raw or semifinished condition. Silica rock produced in one section of the Northwest is almost identical to that quarried in another in handling characteristics, and value f.o.b. cars. The same is true of the various limestone operations. It would appear, therefore, that since little or no cost differential exists between the several movements of silica rock and limestone in the Northwest from the standpoint of handling charges, value, bulk and weight, risk of handling, and value of service to shipper, that the rate for each of these commodities should be determined primarily by the distance hauled. Volume of traffic and periods of movement should have some effect on the tariff although they should represent only a relatively small modification.

Limestone and silica rock are sold f.o.b. cars in the Northwest for two to three dollars per ton, and a freight differential of a dollar or less on such low-cost products is a dominant marketing factor. Variation in freight tariffs, as shown in tables I and II, clearly indicates the competitive position of the various producing localities for the markets.

Although this paper is concerned with limestone and silica rock, the variance in rates for like hauls can be duplicated in other industrial minerals. In general freight rates make up the major part of the cost of producing industrial minerals and therefore the freight rate usually determines whether or not such a project is economic. It is evident that establishment of rates is unscientific and usually on the basis of "all the traffic will bear." This brings into focus the extreme need for reviewing freight rates on industrial minerals to obtain more uniformity in such rates and less power to the railroad to make or break a shipper or prospective shipper.

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Fuel Economy in the Lepol Kilns

by R. A. Kinzie, Jr.

In a conventional cement plant, the drying and heating of the raw material takes place in the upper section of a cylindrical kiln where the heat exchange is poor.

In a Lepol kiln this part of the process is carried out on a traveling grate where the heat exchange is efficient. The increase in fuel economy justifies the greater complexity of the installation.

THE major operating costs in a cement plant are labor, power, and fuel. The opportunities and methods of savings in labor and power parallel other industries. Because our industry's use of fuel is large and its method of use is unique, the devices adapted for saving fuel are of interest.

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In normal practice the raw material in the form of a dry powder or a mud is fed to a rotating inclined cylinder where it is dried, calcined, and clinkered. In a Lepol kiln the raw material is rolled up into little balls with water in the nodulizer, fed to a traveling grate on which it is dried and partially calcined and then to a short kiln where the process is finished.

The more usual methods of saving heat have been the use of long kilns with chains in wet process plants and the use of waste heat boilers in dry plants (fig. 1). At the Santa Cruz plant we have used a Lepol system kiln and achieved a fuel economy of 4.7 gal per bbl of clinker or about 704,000 Btu per bbl. This can be compared with our old kilns using the same fuel and raw material which consume 7.8 gal per bbl or 1,130,000 Btu per bbl.

These are gross Btu and not the net figures usually used in reports of foreign companies.

Our average consumption for 1948 was 708,000 Btu per bbl of Lepol clinker, based on cement sold, and oil purchased. The overall fuel consumption for 1948 was 6.61 gal per bbl.

The Lepol system consists of a nodulizer, a grate, and a kiln. There is the usual accessory equipment which is similar to that used in other cement plants. The following description covers the equipment in operation at our plant.

The nodulizer is a cylinder 8 ft 10 in. in diam by 18 ft long set on a slope of ½ in. per ft, and rotated at about 10 rpm. It is driven by a 50 hp wound rotor motor drawing about 9 kw. The inside of the shell is cleaned by a continuously operating scraper which draws 0.1 kw. The dry raw material is fed to the nodulizer where, being sprayed with water, it rolls into small spheres about 90 pct being between ½ and 1 in. in diam and containing about 15 pct water.

The nodules fall on the traveling grate where the hot gas from the kiln is pulled down through the

a. 11 ft diam by 121 ft long plus 47 ft grate. Moist process.

b. 12 ft diam by 464 ft long. Wet process.

c. 11½ ft diam by 360 ft long. Wet process.

d. 9 ft diam by 314 ft long. Dry process.

e. 10½ ft diam by 250 ft long. Wet process.

Fig. 1—Size comparison of typical wet and dry process kilns with the Lepol grate and kiln.

nodules and grate and then exhausted. The grate is 47 ft 6 in. between centers and in width 12 ft 5 in. between the centers of the outside chains. The open area of the grate is 40 sq ft which is equivalent to a circle 7 ft 2 in. in diam. At present, the grate travels 32 in. per min and carries a 6 in. bed of nodules.

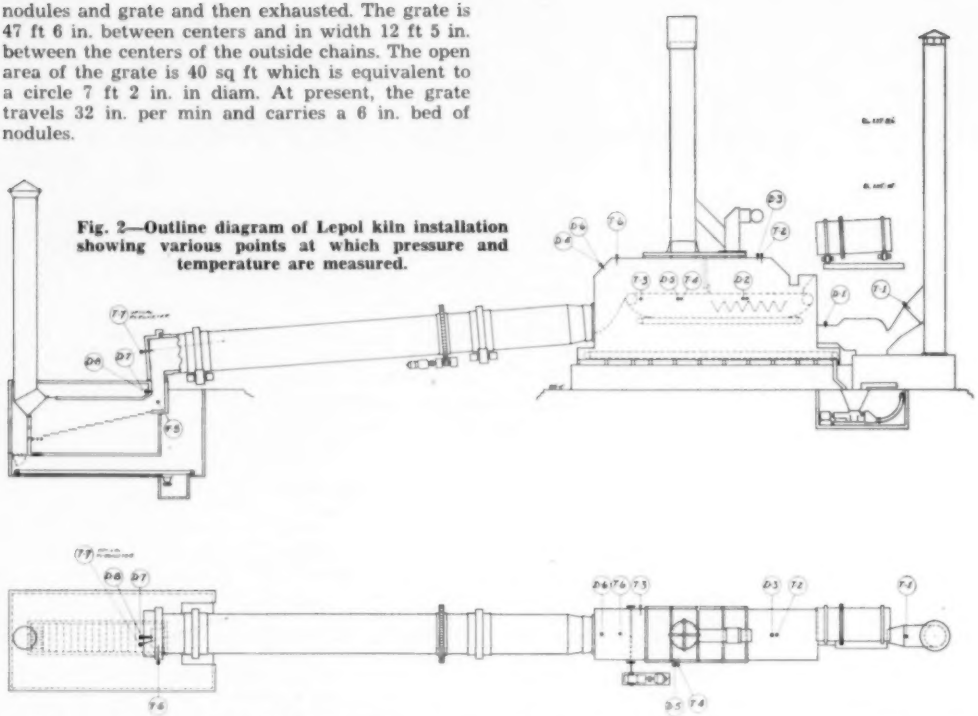

Fig. 2—Outline diagram of Lepol kiln installation showing various points at which pressure and temperature are measured.

The grate is driven by a 15 hp BTA motor drawing 5 kw. On the return run the grate is supported on a motor-driven chain which supports its weight.

The power for this chain is included in the power given for the grate motor.

The material from the grate falls into the kiln of standard design. The kiln is 112 ft long by 11 ft in

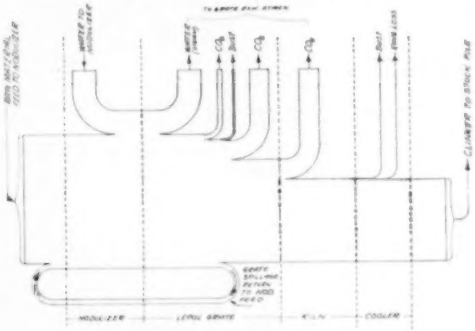

Fig. 3—Material balance, Lepol kiln.

diam supported by two riding rings and driven by a 40 hp wound rotor motor drawing 21 kw. The slope of the kiln is 7/16 in. per ft.

The main fan is driven by a 125 hp motor drawing 76 kw.

The overall power consumption of the nodulizer, grate, kiln, and exhaust fan is 1.25 kw-hr per bbl. This will compare favorably with standard kilns equipped with chains or waste heat boilers.

The kilns do require more labor than normal kilns but this should not exceed $0.03 per bbl of clinker.

The repairs chargeable to the Lepol system are low and the lost time on separate kiln runs has varied from 24 to 42 pct of the total lost time.

The instruments on our Lepol system are greater in number than on a normal kiln. Our old cylindrical kilns have only a thermocouple indicating stack temperature.

On the Lepol the temperatures at the inlet to the exhaust fan, the top of each end of the grate, and the bottom end of the calcining end of the grate are measured by thermocouples and recorded (fig. 2). A ray-o-tube indicates optically and records the burning zone temperature of the rotary kiln. These are all Leeds and Northrup instruments.

Draft gauges indicate the pressure at the inlet to the exhaust fan, the differential through each end of the grate, the differential through the kiln, and the pressure at the kiln hood. The differential through the kiln is used to control the damper at the exhaust fan. The speed of the grate is recorded as well. The speed of the grate is interlocked with the kiln speed and one of three values can be selected by the burner. The nodulizer is independently controlled by the nodulizer operator.

Material Weights: The weight of clinker produced is calculated from the clinker storage measurements adjusted to sales of cement.

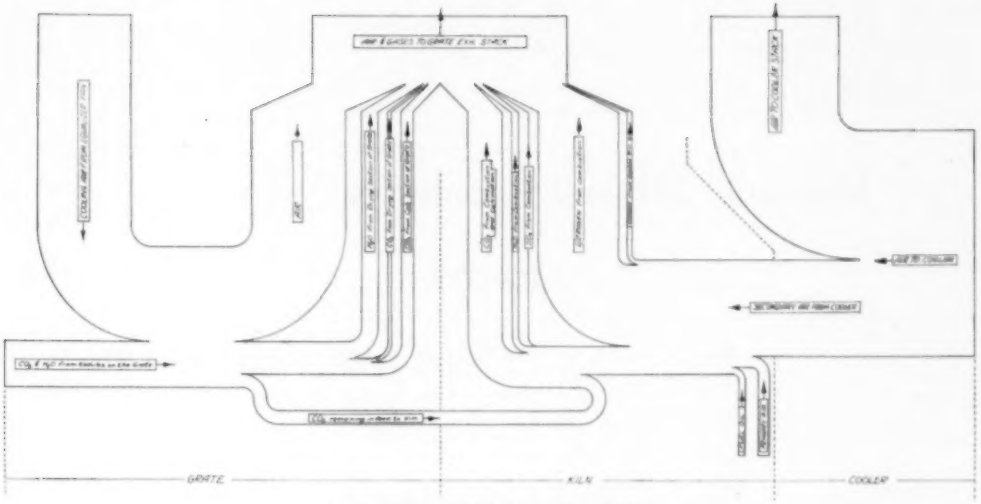

Fig. 4—Gas weight balance, Lepol kiln.

The weights in other parts of the circuit are believed to approximate quite closely the actual values (fig. 3).

Gas Weights: The weights of gas involved have as a basis of calculation the measured volumes in the cooler intake and exhaust, the primary air, and the tempering air drawn into the grate. The volume of gas of combustion is calculated from the analysis and weight of oil burned (fig. 4).

The final weight of air exhausted is a little less than that which has been measured on occasions. No attempt was made to balance the difference by adding a leakage to the calculations.

The weight of oil is an average which agrees within 1½ pct with the average for 1948.

Heat Balance: The values of heat content of the gases and clinker were taken from the "Table of Heat Content" published by the Portland Cement Association. The heat of formation of clinker is that calculated by Martin. The air was assumed to have a 50 pct relative humidity (fig. 5).

Because of difficulty in measuring the gas temperature entering the kiln from the cooler, the heat content was calculated by subtracting the heat leaving the cooler from the heat content of the clinker entering the cooler. The heat leaving the cooler, because of the lower temperatures involved, is easier to measure. The temperature of the clinker entering the cooler was determined by optical means.

The weights of gas involved were taken from calculations illustrated in a previous chart.

The 24,000 Btu unaccounted for includes the loss from the grate housing, the cooler housing, and miscellaneous small losses.

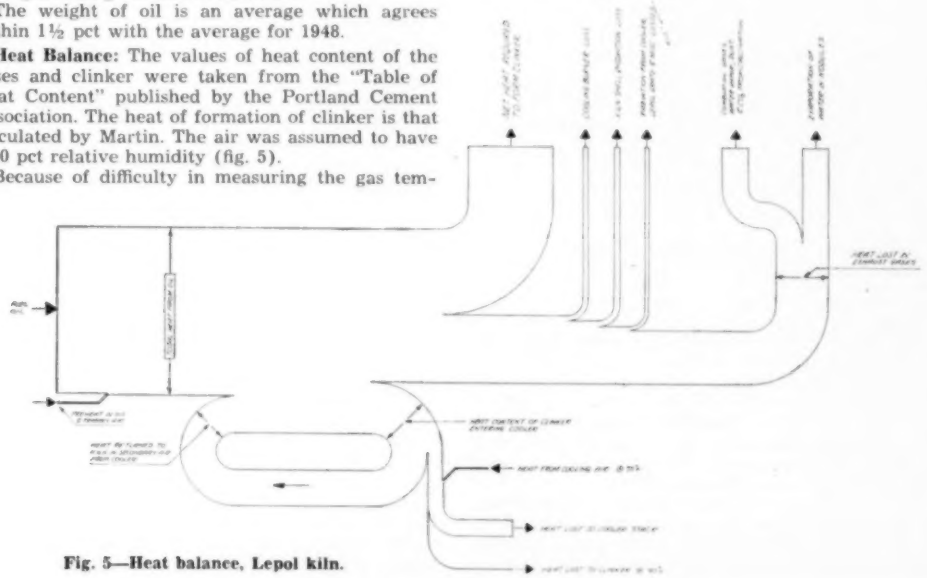

Fig. 5—Heat balance, Lepol kiln.

Industrial Mineral Economics and the Raw Materials Survey

by Raymond B. Ladoo and C. A. Stokes

This paper summarizes the economic problems of the industrial mineral industries which are essentially different from those of the metals and the fuels. Failure to understand and evaluate such factors as markets, place value, and substitutions of one mineral for another may be disastrous. The engineer should be as responsible for profitable operation as he is for technical efficiency. The Raw Materials Survey of Portland, Oregon, is described.

IT is unfortunate that the word "economics" has come to mean, in the minds of many people, a sort of half-baked mixture of New Deal philosophy and bookkeeping. It may mean anything from mine cost keeping to the theory of gold as a monetary standard. As applied to the industrial mineral industries as a whole it means to us an integrated study of all of the technical, geographical, market, financial, and other factors which determine whether or not a given enterprise has a sound chance of making a business success.

Of course it is impossible to generalize too freely in this large group of industries which go to make up the nonmetallic minerals. This group includes such dissimilar products as crushed stone and capacitor mica, phosphate rock and spinning fiber asbestos, roofing granules and potash salts, borate minerals and monumental granite. Yet in many respects their economic problems are similar and, at the same time, far different from those of the metals and of petroleum and coal. These basic differences are almost never recognized by people outside the industries and are rarely appreciated even by metal mining engineers. But in many instances these economic factors are fully as important as, if not more important than, technologic problems.

It has been a long fight to get an intelligent and wide-spread appreciation of sound technology in the nonmetallic mineral industries. The importance of the economic side of the problem has yet to be widely understood and appreciated. The problem was outlined in considerable detail by one of the authors some 18 years ago in a long series of articles* and he has since prepared numerous talks and papers on the subject. But it has seemed like a dry, uninteresting matter. Little has been done to spread the gospel where it is most needed. But it is

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* Raymond B. Ladoo: Economics of the Non-Metallic Mineral Industries. *Rock Products* (Feb. 28, 1931 to June 18, 1932).

as much the duty of the engineer to give sound advice on economic problems as it is on technical problems. This has been recognized in most descriptions of the function of the engineer. Yet that phase of engineering is usually overlooked. Too many men who call themselves engineers have little or no ability to evaluate economic problems. Efforts are now being made to remedy this situation by giving courses in mineral economics in some of our universities. The Division of Minerals Economics at Pennsylvania State College is an outstanding example.

Every industry must be governed by economic considerations. But there is little point in repeating again the well-known and often tabulated economic factors common to most businesses. In this class are such factors as labor supply, efficiency and cost, climatic factors, interest rates, local taxation, and a host of others. Other factors peculiar to the mining industry in general are well known to most engineers and are to be found in books on mine valuation and in handbooks on mining engineering. Except as they have unusual characteristics in the industrial mineral industries they will not be discussed here. It is our purpose to think about, talk about and popularize those factors of minerals economics that are peculiar to the industrial minerals.

Economic Factors Peculiar to the Industrial Minerals: While it is difficult to give short titles properly descriptive of some of these factors we can attempt to list some of them as follows:

1. Lack of free markets.
2. Importance of place value.
3. Variable mineral composition and inaccurate nomenclature.
4. Interchangeability of minerals for some uses.
5. Time required for product and market developments and effect on capital needs.
6. Nonmetallic mineral "alloys," natural and artificial.
7. Synthetic minerals and mineral substitutes.
8. Inadequate application of research and development to production, processing, and application problems.
9. Influence of technologic developments.

These factors are discussed briefly below.

Lack of Free Markets: By this we mean that few industrial minerals are such uniform, standardized articles of commerce that they can be sold and used interchangeably, anywhere, at any time. At one end of the scale of marketability is gold, which can be sold or exchanged freely anywhere in unlimited quantities. At the opposite extreme are many nonmetallic minerals such as feldspar which must have definite consumer acceptance before they can be sold and then only in limited quantities. Few industrial minerals can be used directly by the ultimate consumer. They are not consumer goods. Except by finding new uses total consumption can grow only as industrial users' markets are developed and grow.

Place Value: Many industrial minerals have such low unit value or are so abundant that proximity to markets is more important (with obvious reservations) than quality or quantity. Thus a large deposit of high grade building sand near large and growing industrial centers may be very valuable, while a similar deposit in the middle of a Nevada desert is valueless.

But place value is something which is constantly changing due to a wide variety of factors. Among these are:

1. Changing geographical distribution of population and of industries, bringing formerly worthless deposits within economic shipping radius of consuming industries.

2. Improved production processes may improve quality or reduce costs or both, so as to offset, in part, high transportation costs.

3. Improved transportation methods, from deposits to railroad, make remote deposits increasingly accessible economically. Among these should be noted the airplane (asbestos and jade from interior Alaska), good roads and motor trucks, pipelines (clay in slurry form pumped from deposits to mills in Georgia), and long cross-country hauls of cement raw materials by belt conveyor.

4. New market demands, both for quantity and quality, gradually justify the working of remote deposits.

5. As the chemical and physical processing of minerals becomes more complicated and extensive, plants tend to become larger and energy needs much greater. The future may see a greater trend toward development of minerals near sources of cheap energy.

6. As costs for labor and plant construction in the process industries rise, raw material costs constitute a smaller percentage of total costs. Thus the paying of higher transportation charges on raw materials from remote sources, may become relatively less objectionable.

This list could be extended but enough has been said to demonstrate the great importance of location of deposits with respect to markets.

Variable Mineral Composition and Inaccurate Nomenclature: Among the many factors which make the economics of this field difficult to grasp and understand is the wide variability in composition and chemical properties of many materials which we call by a single mineral name.

While this is true in the whole field of mineralogy it is not so important in the group of metallic ore minerals. Of course, the mineralogy of an ore has an important bearing on the ease and cost of ore beneficiation and of smelting and refining. But the product finally marketed is relatively pure, uniform, and standardized. The salable end products, whether they be copper, lead, zinc, or tin, are standard regardless of the mineralogy of the original ores.

But in the nonmetallic mineral field the names we call our final products may be no guides at all to the chemical composition or the physical and chemical properties. Any number of minerals could be used as examples, asbestos, mica, clay, graphite, bentonite, feldspar, ocher. But as a fairly simple example let us take talc.

The mineralogies give us a definite chemical formula and percentage composition for the pure material, of which, of course, there is very little

produced commercially. The talcs of commerce vary greatly from the pure material, both in chemical composition and in physical properties. The so-called fibrous talcs of St. Lawrence County, New York, differ so greatly from the talcs of Vermont and Georgia that they find almost wholly different uses, they are noncompetitive. In fact, recent careful work by the U. S. Geological Survey on the New York talcs have proved that much of the material sold as talc contains as little as 5 pct talc (or even none at all in one case), the balance being a mixture of tremolite, serpentine, anthophyllite, and occasionally diopside, quartz, calcite, and other minerals.

But U. S. Bureau of Mines statistics of talc production include all material sold as talc, plus finely ground soapstone, plus pyrophyllite. How can a prospective new producer of talc from, say, a newly discovered deposit in Michigan, gain an adequate idea of the size of the potential markets for his talc from published statistics? Worse still, how can he know that the statistics he finds mean very little without experienced interpretation?

Even the average metal mining engineer would not suspect that published production statistics were not a good and safe guide. Otherwise able metal mining engineers have made some very costly blunders when they have, with overconfidence due to ignorance, tried overnight to be experts on industrial minerals. They did not know the problems and they did not know that they did not know.

How can we as industrial mineral engineers and technologists educate the rest of our own profession at least to the realization that our problems are different?

The Interchangeability of Minerals: The interchangeability of minerals or minerals and chemical products, complete or partial, for many important uses is an economic factor, the importance of which is often overlooked even by experienced producers. Consider feldspar. The feldspar producer not only must compete with other feldspar producers but also for many uses, with aplite, nepheline syenite, artificial mixtures such as Gar-Spar, feldspathic sands, and, to some degree with pyrophyllite and talc. This list could be extended. Unless he realizes this situation, a present or prospective feldspar producer may seriously err in his judgment of market possibilities. This type of competition may or may not be very serious.

From the standpoint of a whole industry, feldspar again for example, this substitution of one product for another may not be too serious. The industry as a whole may show a satisfactory rate of growth. This may be due to fortuitous increases in use by present consuming industries, feldspar in glass for example, or to entirely new uses. But the change in geographical distribution of markets may be disastrous to individual producers. Old producers may have to drop out and new ones, more favorably located, take their places.

The extent of this type of competition can here be only touched upon briefly, but it includes competition not only with other minerals, rocks, and mixtures of materials, but also with chemically prepared products. An example is chemically precipitated calcium carbonate as a paper filler and coating material to replace clay.

This interchangeability of materials is at once a danger to established producers and an opportunity, for it may extend and widen markets for their own materials. In any event it further emphasizes the basic need for mineral producers to study markets. It is a challenge to the ingenuity and imaginativeness of all executives and engineers in the mineral industries.

In studying markets it is not enough to know that the A.B.C. Co. today uses 5000 tons of talc per year. We must know what kind of talc, why and how it is used, exactly what physical or chemical function it performs in the user's products or processes. Is there any other mineral or chemical product or mixture of materials which will perform this essential function satisfactorily? Can such a product be obtained anywhere else in adequate supply, of proper quality and uniformity, and at an equal or lower delivered cost?

To obtain and understand this kind of data an engineer must have a good background of industrial chemistry and acquire a fairly detailed knowledge of many industrial processes. Often plant production men, engineers and chemists, have an astonishing lack of knowledge of raw materials and their functions, and the mineral producer himself may have to dig out the information he needs.

Time Required for Product and Market Development and Effect on Capital Needs: Due to variability in composition and lack of free markets for many industrial minerals a new producer must usually spend considerable time in developing methods of production to make uniform products acceptable to the trade. It may take months to open up enough working faces in the mine or quarry to supply daily mill requirements. Even if the final products appear to be identical with established products, consumer acceptance may be slow except in times of shortages.

The result of these delays is that a new project may have to wait two years or five years or even ten years, before there are any profits. To tide over this nonprofit period may require large capital expenditures for production labor, power, supplies, taxes, research and market development, and other items. Many new producers and, often, experienced producers overlook this factor and fail to provide sufficient capital at the start. Too often additional capital is not available when needed and the project, sound enough in itself, fails before it is well started.

In contrast with this, some metal mining and milling operations are so standardized and markets are so well established that a new mill may be built and operating to capacity in from six months to a year.

Nonmetallic Mineral "Alloys," Natural and Artificial: Some industrial minerals useful for their chemical properties or for both chemical and physical properties, such as ceramic raw materials, are mixtures of two or more minerals, somewhat similar to metallic alloys, but the ingredients occur in natural mixtures and in variable proportions. Feldspar is a good example. Practically all commercial feldspars are mixtures of potash, soda and lime feldspars, plus silica and minor accessory minerals, and no two are exactly alike.

Natural cement rock is a mixture of lime, silica,

and alumina minerals in such proportions that, after calcining, the product has the properties of a hydraulic cement. Blast furnace slag used for cement making is an artificial mixture of these same ingredients.

Such mixtures are always of variable composition and physical properties and require special techniques to make uniform, salable products. The production of pure metals, such as gold, silver, copper, lead, and zinc, requires no analogous processing.

Synthetic Minerals and Mineral Substitutes: Chemical processes are used today to make numerous synthetic and artificial products which replace natural mineral products to a greater or lesser extent.

Silicon carbide is a good example of an artificial mineral, not found in nature, which has nearly driven from the market natural abrasives of certain types. Electrically-fused pure aluminum oxide similarly replaces most of the natural corundum and emery formerly produced for abrasive use. These two artificial abrasives together have also made nearly obsolete the production of many types of grindstones, pulp stones, and sharpening stones of all kinds.

Synthetic ruby and sapphire have nearly replaced natural stones for jewel bearings.

Quartz crystals suitable for radio oscillator plates and sheet mica of capacitor grade have been made artificially, but as yet the synthetic products are no threat to the natural minerals.

Artificial graphite has replaced much natural graphite for certain types of uses.

While we are not yet able to make spinning fiber asbestos synthetically, glass fibers are constantly being improved and may in time replace asbestos.

In the metal field, no one has yet been able to make synthetic gold or copper or tin, commercially, although necessity has forced widespread substitutions.

It is of interest to note that most mineral substitutes are themselves made from other minerals.

Inadequate Application of Research and Development to Production, Processing, and Application Problems: While it is true that the last two decades have seen great progress in the application of engineering and technologic principles by the industrial mineral industries, there are still many industries which could profit by more research and development work.

One of the trends in research and development today is the seeking of more specific functionality even at considerably increased costs, in other words, sharp-shooting the consumers' needs. An example of this is the shipping of clays and precipitated calcium carbonate in slurry form in tank cars. Other examples are the pelletizing of dusty fillers and pigments, coprecipitation of pigments and fillers with rubber, and master-batching with plasticizers.

Another tendency related to the above is that toward more chemical and physical modification of minerals. Outstanding examples are the micron size fractionation and the bleaching of clays for paper fillers and coatings. Crude clays are used today which, until recently, were considered worthless for these uses. The addition of surface coatings to pigments and fillers is another example.

Influence of Technologic Developments: Improved methods and equipment for mining, treatment and testing of industrial minerals have economic effects far beyond savings in costs and improvement in quality.

In the past few years there have been many new inventions and technological improvements in processing methods and machines which have had a profound effect on established industries. Some of these are noted below:

1. Froth flotation
2. Sink-and-float or heavy-media separation
3. Greatly improved air separation
4. Centrifuging, either alone or with electrophoresis
5. Micronizer—fluid energy grinding
6. High intensity magnetic separation
7. Improved electrostatic separation
8. Practical bleaching methods for white fillers
9. Spray drying
10. Air conveying
11. Electric eye controls
12. High gravity artificial grinding pebbles
13. Humphreys spiral concentrator.

Other devices could be cited. Many of these, of course, are not limited to the nonmetallic minerals.

These improved working tools lead to the use of lower grade or alternative minerals, thus increasing reserves and widening the geographic distribution of reserves.

We now have new and improved equipment and techniques for examining and testing minerals and mineral products, such as the electron microscope, X ray diffraction techniques, and spectroscopic methods of analysis. These all contribute to a better understanding of the composition and properties of minerals and tell us why certain minerals behave as they do in use applications. With the help of the basic knowledge that these tools contribute, we can find other minerals or combinations of minerals and chemicals as substitutes which may do a given job better or cheaper.

The geographic distribution of consuming industries, of population, and of per capita wealth, due largely to war developments, hydroelectric plant construction as in the T.V.A., Hoover Dam and Bonneville areas, greatly increased freight rates, and the Supreme Court basing point decision on portland cement, all have very great influence on industrial mineral production and markets. These changes have created new consumer markets, leading to the building of localized plants to serve those markets. This, in turn calls for new sources of raw materials, local if possible, but at least available at permissible costs.

Technological advances together with new consuming areas to provide markets make deposits commercially valuable which once were of no interest.

The Raw Materials Survey: Much of the preceding discussion has been upon the subject of markets for mineral products and their influence on the value and workability of deposits. In the past our supplies of mineral products have been sufficiently abundant and low enough in price so that producers have had to seek markets. Until recently, industrial buyers have not felt the need to worry much about sources of raw materials. Market surveys have become commonplace, but only recently have we begun to hear about the reverse of the picture, raw materials surveys.

An outstanding development of this new concept is The Raw Materials Survey, of Portland, Oregon, an organization conceived in 1946 and founded in 1947, to study industrial raw material problems of the Columbia River basin in the Pacific Northwest. It was an outgrowth of special studies of sources of certain mineral raw materials made by the senior author and sponsored by the Industrial Committee of the Portland Chamber of Commerce. The concrete plans for a continuing agency to study such problems were made by Paul B. McKee, president of Portland Gas and Coke Co., Chester K. Sterrett, chairman and manager respectively of the Industries Committee, and the senior author.

This nonprofit organization is supported by Chambers of Commerce, utility companies, steamship and railroad companies and port authorities of the area. Its president is General Thomas M. Robins and the managing engineer is A. O. Bartell.

The purpose of the survey is to gather and furnish to interested companies information on sources of raw materials needed by present and potential industries of the area. The need for this work arose from the fact that there was no single agency which could supply accurate, essential information of this character. It had been sometimes said that while the Pacific Northwest had ample water, electric power, and agricultural and forest product resources, it was deficient in mineral and other raw materials essential to a balanced industrial economy.

One obvious answer to this is that while Connecticut has no copper or zinc deposits, its brass industries have grown and prospered for nearly 200 years. Another answer is that substitute raw materials, locally produced, may replace some conventional raw materials and even perhaps do a better job. A third answer is that foreign raw materials, brought in by water, may possibly be delivered in Portland or Vancouver or Longview more cheaply than from domestic sources. Our eastern seaboard industrial economy in the early days was built largely on foreign trade.

The Raw Materials Survey does not duplicate the work of any other agency. While one of its functions is to make special studies for which it gathers much original data, its daily function is to gather, from every possible source, information on sources, qualities, freight rates and prices on a wide variety of raw materials, mineral and nonmineral, and to study raw material requirements of present and possible future industries. The results of this work are either published as special surveys or in a Monthly Newsletter, or used to answer inquiries.

This work requires knowledge of the requirements of the process industries, strong imagination and resourcefulness, and a keen eye for ferreting out odd bits of seemingly unrelated data and piecing them together.

The underlying policy of this Survey is not to boom local raw materials, good or bad, but to give consuming industries honest, reliable facts as to where they can get essential raw materials. Industries do not care whether the sources are local, domestic, or foreign provided they can get what they want in adequate quantity of proper and uniform quality at a price they can afford to pay.

At times, it has been thought that this policy was detrimental to local and state mineral industries. But this is not a valid criticism. If local industries can produce what is needed on a competitive basis, they will get the business, but there is no point in artificially bolstering up a local weakling just to satisfy local pride. In the long run sound consuming industries are of more value to a state than uneconomic raw material producers.

Stockpiling Problem: An economic problem much in the public eye today is that of stockpiling of minerals essential to war. Here again, the case of most nonmetallic minerals differs from that of the metals. First, most nonmetallic minerals occur in relative abundance in this country, notable exceptions being radio-grade quartz crystals, capacitor-grade mica and mica splittings, industrial diamonds, crucible-grade graphite, and a few others. Second, for many uses there are acceptable or partial substitutes, or by research substitutes can often be found. Contrast this with the metals, where there are only moderate opportunities to develop substitutes for iron, copper, lead, zinc or tin.

For such nonmetallic minerals as we have in fairly adequate supply, the only reason for stockpiling is that war demands may come at a rate much faster than our ability to produce. For really abundant minerals this seems hardly an adequate reason.

For materials like fluorspar, where our reserves are adequate for only a short term and where the rate of extraction is limited by physical factors, stockpiling, if from foreign deposits, has the great additional merit of extending our reserves.

The alternative to the stockpiling of minerals which we cannot produce here, such as quartz crystals, is research to make them synthetically or find adequate substitutes for them. As is well known, such research is now going on. But it cannot be depended upon to solve our problems quickly and stockpiling for military needs is essential.

Conclusions: It should be evident that the engineer and the producers in the industrial minerals field should be as cognizant of economic problems as they are of technical and engineering problems. In most instances, the economic problems are of first importance, because if products cannot be sold at a profit, clever mining or processing is of no avail; the project fails. The engineer is responsible for the overall success or failure of the project. It has been impossible here to give more than a very brief and incomplete outline of the problems, but its importance deserves the thought and attention of many engineers and producers.

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Annual Meeting Memos

Registration Fee—Members, $3., nonmembers, $5., no fee for Student Associates.

Coal

The annual meeting program is detailed here for the first time.

Monday Morning—*Coal Mining*

The Continuous Miner. J. J. Snure.

Problems of Economics and Recovery Involved in Working Several Beds of Coal Simultaneously or One After the Other. H. H. Hasler.

Responsibility of Ownership and Top Management for Underground Maintenance. Ralph M. Hunter.

Monday Afternoon—*Safety*

Prevention and Elimination of Underground Fires. F. J. Peternell.

Present Practices and Future Outlook in Dust Control in the Anthracite Industry. R. Emmet Doherty.

Answers to Specific Safety Problems in West Virginia Mines. A. J. Alexander.

Tuesday Morning—*Fine Coal Cleaning I*

Cleaning of Fine Coal by Tables. Ray E. Zimmerman.

Some Recent Investigations with the Dutch State Mines Cyclone Separator on Fine Coal Slurries. S. A. Falconer.

Cyclone Thickener Applications in the Coal Industry. M. G. Driessen and H. E. Criner.

Tuesday Afternoon—*Fine Coal Cleaning II*

Kerosene Flotation of Bituminous Coal Fines. L. E. Schiffman.

Laboratory Performance Tests of the Humphreys Spiral as a Cleaner of Fine Coal. M. R. Geer, H. F. Yancey, C. L. Allyn and R. H. Eckhouse.

Rheolaveur System of Fine Coal Cleaning. John Griffen.

Coal Preparation

Wednesday Morning

Qualitative Efficiency of Separation of Coal Cleaning Equipment. W. W. Anderson.

Operating Data for a Bird Centrifuge. A. C. Richardson, and Orville R. Lyons.

Drying of Fine Coal. V. F. Parry and E. O. Wagner.

Wednesday Afternoon—*American Men and Machines in Foreign Coal Mining*

England, Holland, China, Turkey, Australia. John Griffen, and J. W. Woomer.

France. W. W. Dartnell.

Poland. H. F. Hebley.

Germany. R. R. Estill.

Chile. Clayton G. Ball.

South America. Thomas Fraser.

South Africa. Byron M. Bird.

India. Roy S. Bigelow.

Mining Methods

Monday Afternoon

Diesel Truck Haulage through Inclined Adit from Mine Stopes to Mill. V. C. Allen.

An Arrangement for Pumping Acid Mine Water at a Tri-State Mine. H. A. Krueger.

Millisecond Delay Detonators. D. M. McFarland.

Pine Creek Tungsten Mine. David D. Baker.

Tuesday Morning

Theory and Practice of Caving. F. S. McNicholas.

Classification of Block Caving Methods. F. S. McNicholas.

Block Caving at Cleveland-Cliffs Iron Co. Mines, Marquette Range. D. K. Campbell and H. H. Korpinen.

Review of Progress in the Caving of Asbestos Ore. Gerald Sherman.

Tuesday Afternoon—*Joint Session, Mining Methods and Industrial Minerals.*

Mining Research. H. S. Mutz.

Failure of a Mine Opening as Observed in a Mine Model. W. E. Bush.

Theoretical Determination of Strength of Mine Roof. Emery Sipprelle.

Mining Methods and Shaft Sinking at Johnson Camp, Dragoon, Arizona. Roy W. Moore.

Rock Hardness as a Factor in Drilling Problems. W. B. Mather.

Wednesday Morning—*Joint session, Mining Methods and Mining Geology.*

Geological Mapping of Stopes and Its Application to Mining. Thomas E. Gillingham and F. H. Howell.

Geology Applied to Mining in the Picher Mining Field. Joseph P. Lyden.

Integration and Correlation of a Geological Staff and Geological Work in a Mining Organization. H. J. Fraser, J. D. McAuliffe and D. R. Lochhead.

The Function of a Mine Geological Department. J. D. Bateman.

Wednesday Afternoon—*Joint session, Mining Methods, Mining Geology and Industrial Minerals.*

Mine and Surface Maps. Neil Arthur O'Donnell.

Relationship of Geology to Underground Mining Methods. George B. Clark.

Granite Quarrying and Technology. F. S. Eaton.

Ores and Mining in the Itabira Iron District, Brazil. John Van N. Dorr, IV, and Gilbert Whitehead.

Thursday Morning—*Division Business Session—Miners, Geologists, Geophysicists.*

Clean-up session for technical discussions cut short during previous sessions. Presentation of plans, policies and constructive criticisms. Open-house for ideas.

AIME Annual Meeting Program, February 12-16, 1950

Statler Hotel, New York

Saturday, February 11

10 a.m. to 5 p.m.—Council of Section Delegates.

Sunday, February 12

9:30 a.m.-12 m.—Inst. of Met. Div. *Review of Solidification.*
10 a.m.—Board of Directors.

1:30 p.m.—Institute of Metals Division.
2 p.m.—Board of Directors.
3 p.m.—Mineral Industry Education Division. *Columbia University.*

6:45 p.m.—Mineral Industry Education Division. *Columbia University.*
8-10 p.m.—Inst. of Met. Div.

Monday, February 13

8:30 a.m.—Registration.
9 a.m. to 12 m.—Committee on Comminution Research.
Institute of Metals Division *Alloy Systems: Nitrogen and Carbon in Steel.*
Institute of Metals Division *Symposium on Tube Production Practice.*
Extractive Metallurgy Division *Aluminum, Primary and Secondary-Magnesium.*

9:45 a.m. to 12 m.

Coal Division *Mining.*
Mining Geology and Geophysics
Minerals Beneficiation Division *Flotation.*
Mineral Industry Education Division.

12:15 p.m.—Welcoming Luncheon. Speaker, Dwight D. Eisenhower.

2:30 p.m.

Mineral Economics
Mining Methods
Mining Geology and Geophysics
Coal Division *Safety.*
Industrial Minerals Division
Petroleum Branch *(Technology).*
Minerals Beneficiation Division *Crushing and Grinding.*
Extractive Metallurgy Division *Titanium-Zirconium-Manganese.*
Institute of Metals Division *Alloy Systems: Structure and Stability.*
Institute of Metals Division *Symposium on Tube Production Practice.*

4 p.m.—Annual Business Meeting.
4 p.m.—Institute of Metals Division Membership Committee.
5 p.m.—Executive Committee, Coal Division.
5 p.m.—Directors' Meeting *Executive.*

Evening—Dinner-Smoker.
8 p.m.—Student Meeting.
Student Careers Conference and Beer Party.

Tuesday, February 14

8:30 to 9 a.m.—Meeting of presiding officers and authors of the day.

9:00 a.m. to 12 m.

Mining Methods.
Minerals Beneficiation Division *Symposium: Mill Management.*
Mining Geology and S.E.G.
Coal Division *Fine Coal Cleaning—I.*
Industrial Minerals Division.
Petroleum Branch *Seminar: Petroleum Economics and Technology.*
Extractive Metallurgy Division *Copper.*
Inst. of Met. Div. *Alloy Systems: New Alloys.*
Iron and Steel Division *Gen'l Session.*
Woman's Auxiliary *Annual Meeting.*

12:15—Luncheons

Minerals Beneficiation Division.
Industrial Minerals Division.
Coal Division.
Mineral Industry Education Division.
Executive Committee, Institute of Metals Division.
Mining and Metallurgical Society of America.

2 p.m.

Minerals Beneficiation Division *Symposium: Mill Management.*
Mining Methods and Industrial Minerals.
Mining Geology and S.E.G.
Coal Division *Fine Coal Cleaning—II.*
Petroleum Branch *Production Review.*
Extractive Metallurgy Division *Zinc.*
Inst. of Met. Div. *Mechanical Properties: Theory and Light Metals.*
Iron and Steel Division *Metallurgical Slags.*
Mining and Metallurgical Society of America, 2 to 4 p.m.

4:00 p.m. AIME Nominating Comm.
4 to 6 p.m.—AIME Research Comm.
5:15 p.m.—Executive Committee, Industrial Minerals Division.

Evening

Metals Branch Dinner (IMD, ISD and EMD).
Petroleum Branch Dinner.
Informal Dance.
Columbia University Alumni Dinner.
Missouri School of Mines and Met. Alumni Dinner.

Wednesday, February 15

8 to 9:15 a.m.—"Scotch Breakfast," Minerals Beneficiation Division.
8:30 to 9 a.m.—Meeting of presiding officers and authors of the day.

9 a.m. to 12 m.

Mining Geology and Mining Methods.
MBD—*Operating Controls.*
Minerals Beneficiation and Industrial Minerals Division *Flotation.*
Coal Division *Preparation.*
Industrial Minerals Division.
Petroleum Branch *General Meeting of Executive Committee.*
Institute of Metals Division *Mechanical Properties: Steel,* 9-10:30 a.m.
Iron and Steel Division, 9-10:45 a.m. *Physical Chemistry of Liquid Iron.*
Extractive Metallurgy Division *Symposium: Fume and Dust Collection.*

10 a.m.—Woman's Auxiliary *Round Table.*
10:30 a.m.—Institute of Metals Division *General Business Meeting.*

11 a.m.—Institute of Metals Lecture by E. E. Schumacher.

12:15 p.m.—Luncheons

Mining Methods.
Mining Geology. Governor Clinton Hotel.
Executive Committee, Extractive Metallurgy Division.
Committee on Physical Chemistry of Steelmaking.

2 p.m.

Mineral Economics.
Mining Geology, Mining Methods
Industrial Minerals. Governor Clinton Hotel.
MBD—*Cyanidation.*
MBD—*Flotation.*
Coal Division *American Men and Machines in Foreign Coal Mining.*
Petroleum Branch *General Meeting of Executive Committee.*
EMD—*Symposium: Fume and Dust Collection.*
Inst. of Met. Div. *Properties: Effect of Alloy Elements.*
Institute of Metals Division *Powder Metallurgy.*
Iron and Steel Division *Physical Chemistry of Liquid Iron.*

4:30 p.m.—Mining Branch Meeting.
4.45 p.m.—EMD—*General Meeting.*

Annual Banquet and President's Reception.

Thursday, February 16

9 a.m. to 12 m.

MBD—*Materials Handling.*
Industrial Minerals Division *Clays and Fillers.*
EMD—*Lead-Antimony.*
Inst. of Met. Div. 9-10:45 a.m. *Properties: Elasticity and Anelasticity.*
Institute of Metals Division, 9-10:45 a.m. *Powder Metallurgy.*
Iron and Steel Division *Bessemer Converter Operations,* 9-10:45 a.m.

10 a.m.—Mining Methods, Mining Geology, Geophysics *Joint Session.*
11 a.m.—Howe Memorial Lecture *by Francis B. Foley.*

12:15 Luncheons

Executive Committee, Iron and Steel Division.
Powder Metallurgy Committee.

2 p.m.

MBD—*Clean-up Session.*
EMD—*Cobalt-Nickel-Tin.*
Institute of Metals Division *Kinetics.*
Iron and Steel Division *Converter Process Developments.*
2:30 p.m.—Inst. of Met. Div. *Powder Met.*

Directors Face Deficit Problem at Dec. 7 Meeting

NO Board meeting being scheduled between that on Nov. 16 and that on Feb. 12 next, the Executive and Finance Committees met on Dec. 7 and Jan. 20 to consider routine matters. At the Dec. 7 meeting, with President Young presiding, the problem of how to meet the 1949 deficit was considered. It was finally voted to ask the Rocky Mountain Fund Committee for $7500 to finance the cost of Vol. 181 of the Transactions, covering metal mining, industrial minerals, mineral industry education, and geophysics; and to ask the Robert C. Gemmell Memorial Fund Committee for $6000 to cover the cost of publishing Vol. 183, on mineral beneficiation. The respective committees subsequently agreed to this use of accumulated interest in the funds, and the volumes will be issued under an imprint to credit the sponsors. The sum of $15,000 was appropriated from the Life Membership Fund, which is a portion of $59,000 that was set aside from surplus during the period 1927 to 1934 to finance service to Life Members of the Institute. Life Members now number about 190 compared with some 390 at that time, so less income is required to meet the expense of supplying them with periodicals and carrying them on the rolls. The remainder of the deficit will be met out of the residue of some $46,000 left in the Special Fund for Publications, contributed some years ago to meet publication costs during periods of reduced income. The current levy on the fund will practically extinguish it for the time being until surplus can again be directed to its replenishment.

Authority was voted to engage an advertising salesman for the "Journal of Metals" and "Mining Engineering" in the territory centering in Chicago, and Joseph Halloran was subsequently engaged for the job.

Besides President Young, those present at the meeting included Donald H. McLaughlin, E. E. Schumacher, Erle V. Daveler, Clyde E. Weed, Andrew Fletcher, H. A. Maloney, and Edward H. Robie.

COMING EVENTS

FEBRUARY

1 **Chicago Section, AIME. H. W. Johnson and C. D. King on prospective developments in steelmaking.**
2-4 Colorado Mining Association, Shirley-Savoy Hotel, Denver.
3 **Eastern Section, Open Hearth Committee, Iron and Steel Division, Warwick Hotel, Philadelphia.**
3 **Columbia Section, AIME.**
6 **Boston Section, AIME.**
7 **Washington, D. C., Section, AIME.**
7 Society for Applied Spectroscopy, New York City.
8 **El Paso Metals Section, AIME.**
8 **San Francisco Section, AIME.**
9 Bituminous Coal Research, annual meeting, Netherland Plaza Hotel, Cincinnati.
10 **Rio de Janeiro Section, AIME.**
10 **Southwestern Section, Open Hearth Committee, Iron and Steel Division, St. Louis, Mo.**
12-16 **Annual Meeting, AIME, Statler Hotel, New York City.**
14 **Delta Section, AIME.**
14 **East Texas Section, AIME.**
15 **Southwest Texas Section, AIME.**
15 **Western Section, Open Hearth Committee, Iron and Steel Division, Roger Young Auditorium, Los Angeles.**
16 **Carlsbad Potash Section, AIME.**
16 **Utah Section, AIME.**
17 **Oregon Section, AIME.**
20-24 ASTM, Shoreham, Washington, D. C.
20 **Detroit Section, AIME.**
21 **Gulf Coast Section, AIME.**
27 **Alaska Section, AIME.**
27-Mar. 3 ASTM committee week and spring meeting, Pittsburgh, Pa.
28 **Montana Section, AIME.**

MARCH

1 **Chicago Section, AIME. F. N. Rhines on recent developments in powder metallurgy.**
3 **Columbia Section, AIME.**
6 **Boston Section, AIME.**
7 **Washington, D. C., Section, AIME.**
8 **Chicago Section, Open Hearth Committee, Iron and Steel Division, all-day meeting, Delprado Hotel, Chicago.**
8 **El Paso Metals Section, AIME.**
8 **San Francisco Section, AIME.**
8-10 API, Adolphus Hotel, Dallas.
10 **Rio de Janeiro Section, AIME.**
14 **Delta Section, AIME.**
14 **East Texas Section, AIME.**
15 **Southwest Texas Section, AIME.**
15 **Western Section, Open Hearth Committee, Iron and Steel Division, Roger Young Auditorium, Los Angeles.**
16 **Carlsbad Potash Section, AIME.**
16 **Utah Section, AIME.**
23-25 Dept. of Geology, Univ. of Kentucky, and Kentucky Geological Survey, symposium on minerals in southeastern U. S., Univ. of Kentucky, Lexington.
29-31 API, Skirvin Hotel, Oklahoma City, Okla.

APRIL

4-7 Nat'l. Assn. of Corrosion Engineers, St. Louis.
10-12 **Open Hearth Conference, and Blast Furnace, Coke Oven and Raw Materials Conference, Netherland Plaza Hotel, Cincinnati.**
19-21 ASCE, spring meeting, Los Angeles.
21-22 **New England Regional Conference, Institute of Metals Divisional, AIME, Hotel Biltmore, Providence, R. I.**
23-26 American Ceramic Society, annual meeting, New York.
24-26 AMC Coal Convention, Netherland Plaza Hotel, Cincinnati.
25-26 Annual Metal Powder Show, Book-Cadillac Hotel, Detroit.

JUNE

6-17 Second Mechanical Handling exhibition and convention, Olympia, London.
24-30 First international conference on coal preparation under direction of Charbonnage du France.
26-30 ASTM 53rd annual meeting and 9th exhibit of testing apparatus and equipment, Atlantic City, N. J.

JULY

3-8 Second Conference on Oil Shale and Cannel Coal, Glasgow.
10-15 World Power Conference, London.

AUGUST

28-31 American Mining Congress, metal mining convention and exposition, Fair Grounds, Salt Lake City.

OCTOBER

4-6 **Petroleum Branch, AIME, New Orleans.**
9-13 **Washington, D.C., Section & Mineral Economics Division, AIME, joint meeting, Shoreham, Washington.**
13 **Southwestern Section, Open Hearth Committee, Iron and Steel Division.**
20 **Eastern Section, Open Hearth Committee, Iron and Steel Division, annual all-day fall meeting, Warwick Hotel, Philadelphia.**
21-27 National Metal Congress, Chicago.
23-25 **Institute of Metals Division, AIME, fall meeting, Hotel Sheraton, Chicago.**
27 **Southern Ohio Section, Open Hearth Committee, Iron and Steel Division, annual all-day meeting, Deshler-Wallick Hotel, Columbus.**

NOVEMBER

3 **Pittsburgh Section, Open Hearth Committee, Iron and Steel Division and Pittsburgh Section, AIME, annual all-day meeting, William Penn Hotel, Pittsburgh.**

DECEMBER

7-9 **Electric Furnace Steel Conference, Iron and Steel Division, Hotel William Penn, Pittsburgh.**

APRIL 1951

2-4 **Open Hearth and Blast Furnace, Coke Ovens and Raw Materials Conference, Iron and Steel Division, Statler Hotel, Cleveland.**

The *Drift* of *Things*

as followed by *Edward H. Robie*

Effect of Dues on Membership

Many AIME members have expressed an interest in whether or not the increased dues have been the cause of an abnormal number of resignations. It is, of course, too early to say with any definiteness, for as this is written, early in January, many members have only just received their bills for 1950, and a considerable number do not normally pay them for a month or two. They would not be likely to resign until faced with the stark necessity of writing out a check. Still, the data available at the end of the year show no cause for alarm, and certainly do not bear out the ideas expressed by the prophets of doom last spring who warned that the loss of members would more than balance any increase of income through the raising of dues.

To establish whether or not something is abnormal, one must first know what is normal, so let us go back to an average for the years 1946-1948 inclusive. In those three years, 0.9 percent of the enrolled members died; 1.1 percent resigned in good standing; and 2.3 percent were dropped for nonpayment of dues. In 1949, faced with an increase in the dues for the next three years, just exactly the same percentages died, resigned, and were dropped for arrears in dues. Without any unusual membership campaign, AIME membership—and all these data are exclusive of Student Associates—showed a greater net gain last year than it did in 1948, the respective figures being 687 and 580.

We have seen many of the letters of resignation, and the bulk of them seem to be in two categories: (1) those whose interests are no longer in the field of the Institute, or only partly so; and (2) members in foreign countries who have to pay what are often excessive sums in their own currency to secure American dollars. The Directors of the Institute are sympathetic to this latter group, and if any use could be made of foreign currencies would be glad to allow dues to be deposited in foreign countries, but unfortunately all the bills that the Institute incurs must be paid in American dollars.

In the data previously quoted, it is notable that more members are suspended for nonpayment of dues than resign. Many times these members later apply for reinstatement and are chagrined to find that they must pay up their arrears, or up to $20 thereof, in order to be reinstated. They perhaps regard AIME membership like a magazine subscription—if you don't want it any longer you just stop paying for it and ignore the numerous letters you receive from the publisher. The AIME is different. Members are in good standing for three months after their dues are payable—or until March 31. In that time they can resign, and should they later wish to apply for reinstatement there will be no new initiation fee or any arrears in dues to pay. But if Institute headquarters does not hear from you we assume you are on a safari, continue to keep your name on the rolls, print it in the Directory, and try to print enough extra copies of your favorite journal so that you can have the back numbers if you wish, when your check finally arrives. Members are not dropped until they have been in arrears for two years, and many pay their arrears in the second year.

Professional Interests

Within the last two or three months all AIME members (exclusive of Student Associates) have received a card asking them to check (1) their major professional interest as expressed by the activities of one of the ten Divisions of the Institute; and (2) all Divisions in which they may have an interest. Some 9000 replies have now been tabulated, which represents more than half of the Institute membership, so should be a good sample, and we hasten to give readers the results to date:

Major Technical Interest

Division	Number	Percent of AIME	Percent of Branch
Mining, Geology, and Geophysics	2380	26.6	57.5
Minerals Beneficiation	605	6.8	14.7
Coal	816	9.1	19.6
Industrial Minerals	339	3.8	8.2
MINING BRANCH		46.3	100.0
Extractive Metallurgy	540	6.0	22.8
Iron and Steel	734	8.2	31.2
Institute of Metals (physical met.)	1083	12.1	46.0
METALS BRANCH		26.3	100.0
Petroleum (Division and Branch)	2251	25.2	100.0
Mineral Economics (no Branch affil.)	138	1.5	
Mineral Ind. Education (no Branch)	56	0.6	
Total	8942	99.9	

Desired Divisional Affiliations

Division	Number	Percent of Total Selections	Percent of Members Replying
Mining, Geology, and Geophysics	4213	20.3	47.1
Minerals Beneficiation	2274	11.0	25.4
Coal	1533	7.4	17.1
Industrial Minerals	2039	9.8	22.8
MINING BRANCH		48.5	
Extractive Metallurgy	1835	8.8	20.5
Iron and Steel	1726	8.3	19.3
Institute of Metals	1978	9.5	22.1
METALS BRANCH		26.6	
Petroleum	2769	13.3	31.0
Mineral Economics	1527	7.4	17.1
Mineral Industry Education	855	4.1	9.6
	20,749	99.9	

The foregoing data are based on returns from 55.2 percent of the Institute membership that had been received when the survey was made, so by simple arithmetic the officers of every Division can figure out just about how many AIME members are interested in their Division. For instance, 6.8 percent of Institute members are primarily interested in mineral beneficiation, but 25.4 percent are more or less interested. On the other hand, 25.2 percent of the membership is primarily interested in petroleum, but only 31.0 percent have any interest at all in that field, which would be expected.

New Members

We fear this department is getting pretty statistical this month, but in connection with the foregoing data the Branch affiliation of those who applied for membership in 1949 is of interest. The number of applications for membership received from those who had not previously been members (excluding new Student Associates but including changes of status of Student Associate to regular grades of membership), plus reinstatements, was as follows:

Branch	Total	Percent
Mining	471	41.4
Metals	262	23.1
Petroleum	404	35.5
Total	1137	

Comparing the professional interests of the present total membership with those of the new applications, it is evident that the Petroleum Branch is growing much more rapidly than either the Mining or Metals Branches. Possible reasons for this are: (1) the petroleum industry is growing faster than the other branches of the mineral industry; (2) the AIME has less competition in the service it gives to the petroleum production industry than it has in the other fields; (3) petroleum men have more money than miners or metallurgists; (4) the service the AIME gives to members of the Petroleum Branch is more satisfactory than its service to the other Branches; or (5) the Petroleum Branch is more actively out for new members than the other groups. Whatever the reason, if the trend continues, the Petroleum Branch will before long boast of a third instead of a quarter of the Institute membership.

A Job Well Done

Especially in view of what has just been said, it is with sincere regret that we must accept the resignation of William H. Strang as Executive Secretary of the Petroleum Division and Branch, Business Manager of the Journal of Petroleum Technology, and Assistant Secretary of the AIME. For much of the progress made by the Petroleum Branch, Bill has been responsible. Further, his relations with Institute headquarters have always been cordial and he has done much to advance the interests of the Institute as a whole. It is good to know that it was not because of dissatisfaction that he left the AIME to join the production division of the American Petroleum Institute in Dallas—it was just that he was offered a much better paid job. His new office will be in the same building as his present one.

Newell Appleton Office Manager

A further step to promote efficiency in the New York office of the Institute was made at the beginning of the year in the naming of H. Newell Appleton as Office Manager. Thus is another of the recommendations of the Johnson Committee, made two years ago, being carried out. Mr. Appleton is thoroughly experienced in his new work, and is well known to a large number of AIME members who have seen him in action at many meetings, particularly annual meetings and those of the Metals Branch. He will have complete charge of the service departments at Institute headquarters, including purchasing, addressograph, receiving and shipping, orders, filing, change of address, membership, and the reception desk. In addition he will be convention manager and will have various staff duties in coordinating the work and promoting the efficiency of the line departments including the Secretary's office, the publications and accounting departments, and the work of the Mining, Metals, and Petroleum Branches. Before making the appointment the Secretary studied three volumes and numerous pamphlets on office management and we gather that it is a profession, like mining engineering, and one that has progressed rapidly in recent years. We have now turned these books over to Mr. Appleton, confident that he will fully live up to his new responsibilities. Which latter word reminds us of the nifty about the man who was applying for a job. His potential employer said to him, "For this job we want a responsible man." "That's me," replied the applicant. "Everywhere I've worked, when something went wrong, they told me that I was responsible."

Among the Student Associates

Fare at **Missouri School of Mines** Student Chapter meetings must be good; 125 men turned out for the Nov. 22 meeting. President Humphrey opened the meeting, got approval for the minutes of the last one and settled back with the rest of the crowd to hear M. E. Benoit report on the MSPE meeting held in St. Louis. D. R. Schooler, of the mining department, spoke briefly on Denver Equipment awards and presented the annual awards. Speaker of the evening, J. E. Weber, head of the Doe Run Division of St. Joseph Lead, gave a talk on the lead mining industry and showed a film on St. Joe mining methods. Attendance at the Dec. 7 meeting zoomed to 181. That evening, C. B. Wheeler, from the St. Louis office of Shell Oil, showed two films, one on prospecting for oil, and the other on drilling a well. Then he told the men how to look for a job. His talk was of special interest to the graduating seniors and resulted in a detailed question and answer session. . . . In November the **St. Louis University** Student Chapter visited St. Joe lead mines in St. Francois, Mo., about seventy miles from St. Louis. They spent the day underground, viewing one of the largest lead mines in the world.

A Job Well Done

On a recent visit to the **University of West Virginia,** we were very much impressed by the activities of the Mining Society. The meetings of this group are a required part of the curriculum for a degree in the fields of mineral engineering which are taught at that university. The society is divided into four divisions, each of which has one meeting a week. Officers are elected for the divisions, and an over-all set of senior officers is elected for the entire society. The purpose of these meetings is to teach the students to talk before an audience and to give them practice in presenting technical papers. Speakers are designated for each meeting and are allotted seven to ten minutes to present their papers. Other students are appointed to discuss the papers. Topics for the talks are taken from the technical journals. Frequent opportunities for each student to speak occur during the semester as there are only 20 to 25 students in each division. This program is required for the junior and senior students, and although they must be members of the Mining Society, they are not required to be Student Associates of AIME although many of them have joined. The Mining Society is under the direction of Professor Laird, but faculty supervision is kept to a minimum during the meetings. Once a month, all of the engineers of the university gather for an all-professional society meeting. Over 800 attend this affair to which a well-known professional engineer is invited to give the address. The talk is usually of general interest to all engineers and frequently deals with engineers' registration and other aspects of professional life. The Mining Society also has a banquet each year. From talking to the students, we gather that this program is exceedingly popular.—JVB

Prize Paper Contest Winners

Prize Paper Contest winners for 1949 have been selected by the judges of the Subcommittee on Student Prize Papers. John A. Harrison came out on top in the graduate class with his paper, Progress in Mechanization in Underground Coal Mines, and wins a check for $100. A check for $50 goes to L. R. Kalman for his paper, Diffusion in Iron and Steel, A Literature Survey.

Of the undergraduate papers submitted, the three judged the best are: The Christensen Sillimanite Deposit Southeast of Dillon, Mont., by Everett O. Bracken, submitted by the Montana Section; The Effect of Copper and Nickel Upon Rate of Sulphur Transfer, by William H. Humphries, submitted by the Pittsburgh Section; and Sodium Sulphate, a Possible New Utah Industry, by Horace K. Thurber, submitted by the Utah Section. Each of the men will receive $100; if they can be at the Annual Meeting, the awards will be given at the All-Institute luncheon; if not, checks will be mailed to them.

Annual Oyster Roast, Washington D.C. Section

An event of the year for the Washington, D. C., Section is its annual Oyster Roast, to be held this year at the National Press Club on Feb. 21. All AIME members in Washington on that date are invited. Write Julian Feiss or Robert B. McCormick for reservations.

June—Paris—Coal Preparation

An international conference on coal preparation will meet in Paris, France, from June 26 to 30. Organized by Charbonnages de France, which operates all coal mines located in metropolitan France, and sponsored by the French Government, the conference will include all types of coal preparation processes, wet as well as dry, and flotation. The program will be mailed upon request to the Secretary of the Coal Preparation Conference, 35 rue Saint Dominique, Paris 7. After the conference delegates will tour the most recent coal washing plants of the French mines. Paris is a popular spot, especially in June, so register early if you want to be assured of a place to stay.

October—Washington—Economic Trends

The Washington, D. C., Section, AIME, is arranging a meeting at the Shoreham Hotel in Washington during the week of Oct. 9, with a program sponsored by the Mineral Economics Division, to analyze economic trends as they affect the mineral industry both at home and abroad. One half of each day will be assigned to general sessions at which a subject of wide interest to all Institute Divisions will be presented and discussed. The other half of the program will probably be divided into several meetings at which more technical papers will be presented. P. G. Spilsbury is chairman of the Washington, D. C., Section, Charles H. Behre, Jr., chairman of the Mineral Economics Division, and S. Power Warren is general meeting chairman.

The met lab at the University of Wisconsin proved a bit dark for pictures, but if you look hard you can make out P. C. Rosenthal way over on the left. Next to him is R. W. Heine, then L. F. Porter, G. J. Barker, Bob Heushen, M. O. Withen, L. E. Young, E. R. Shorey, O. L. Kowalke, and W. J. Rundle. This meeting was reported in ME for January.

AIME Personals

P. K. Aldous left Potrerillos, Chile, where he is assistant concentrator superintendent for the Andes Copper Mining Co., in November for a four month vacation in England.

George W. Allen, who had been chief engineer for Oglebay Norton & Co., Otsego, W. Va., has taken a post with the Republic Steel Corp. at Uniontown, Pa.

Henry B. Allen, executive vice-president and secretary of the Franklin Institute of the State of Pennsylvania, received the degree of doctor of engineering, honoris causa, from Drexel Institute of Technology at graduation exercises on Dec. 13.

Donald J. Brammer is working for the Shell Oil Co., Tulsa, as a junior exploitation engineer. He finished his studies at the University of Minnesota last June.

Edward H. Brinley, since graduating from the Colorado School of Mines, has been working as a field geologist for the Anaconda Copper Mining Co. in Nevada and California. When Mr. Brinley wrote to us in December he, his wife, and son Dana Gordon, born Sept. 10, were in the process of a big move to Chile where he is assistant geologist with the Andes Copper Mining Co. at Potrerillos.

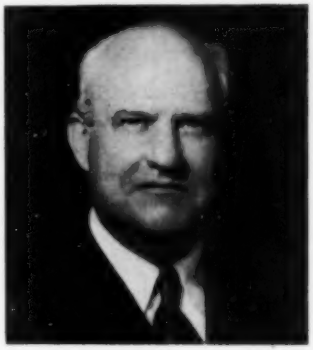
Joseph F. Brown is working in the engineering department of the St. Joseph Lead Co. at Balmat, N. Y.

Arthur H. Bunker, president and director of the Climax Molybdenum Co., has been made a director of the Commercial National Bank and Trust Co. of New York. He is also a director of the American Metal Co., the Firth Steel and Carbide Corp., and the Lehman Corp.

John D. Burgess has returned to this country from Ontario where he was resident mine manager of the Hoyle Mine, a Ventures, Ltd., subsidiary. He has taken a post in the exploration department of AS&R with headquarters in Salt Lake City.

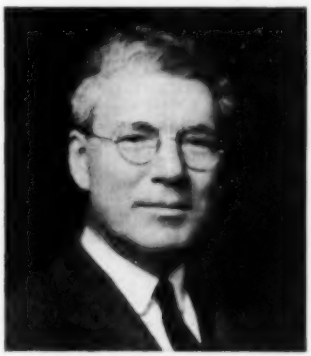
C. M. Campbell, Jr., has been made mine manager for the Camac Mining Co. which is developing property on Vancouver Island, Canada.

Arthur A. Center, consulting engineer of San Francisco, has been given a certificate by the State of California as a professional engineer in chemical engineering.



Howland Bancroft

Howland Bancroft has returned to his New York headquarters after spending the past seven months in South Africa investigating developments in the gold fields of the Orange Free State.



Cadwallader Evans, Jr.

Cadwallader Evans, Jr., resigned as president of the Hudson Coal Co. on Jan. 1. Well known in the anthracite industry, he went to that region during World War I, having had operating experience in soft coal mines near Pittsburgh and in the coal mines of Nova Scotia. He worked for Hudson Coal, then known as the Delaware & Hudson Co., for a couple of years, put in three more with International Salt, and returned to the anthracite region about 1921 to install and operate the first full size Chance Cone Cleaner in the region. He rejoined Hudson Coal in 1925 as general manager.

Douglas Donald paid us a visit at headquarters on Jan. 2 to get acquainted and do a little verbal mining. Mr. Donald is working as a mining engineer for the Annville Stone Co., Hershey, Pa., helping to change over mining from room-and-pillar to shrinkage stopping with long hole blasting. He mentioned that the golf courses in that area are excellent, if you can hit the ball 250 yds. every time.

J. E. Despins has been made branch manager of the El Paso, Texas, office of the Allis-Chalmers Mfg. Co.

J. McLaren Forbes has moved from Silver City, N. Mex., and can be reached at Frontino Gold Mines, Ltd., Segovia, Antioquia, Colombia.

R. M. Fleming has retired from active management and is now living at 517 Pine St., State College, Pa., where he expects to continue his study of the history of bituminous coal mining.

Roy A. Hardy has accepted the chairmanship of the program committee for the 1950 metal mining convention and exposition of the American Mining Congress to be held in Salt Lake City Aug. 28 to 31.

Joseph H. Hedges has been transferred from the superintendency of the Bureau of Mines' Southwest experiment station at Tucson, Ariz., to Washington, D. C., where he heads the minerals division, one of the five main divisions being set up by the Bureau in its reorganization plan.

Robert M. Hernon is now in active charge of the U.S. Geological Survey's metalliferous work in the Silver City, N. Mex., region. His address is Box 773, Silver City. He replaces **S. G. Lasky,** who has been transferred to the Washington offices of the Survey.

Clifford F. Hood, president of the American Steel & Wire Co., Cleveland, became president of another U.S. Steel subsidiary, the Carnegie-Illinois Steel Corp., on Jan. 1.

D. F. Irvin closed his long-time and satisfying association with Oliver United Filters Inc. in different parts of the world on Jan. 1 and is now available for limited engagements on engineering matters related to his past experience. He can be reached at his home, 250 Bronxville Rd., Bronxville, N. Y.

What Went on at Recent Local Section Meetings

SECTION	DATE	ATTEND-ANCE	SPEAKER, AFFILIATION, AND SUBJECT
Ajo Subsection	Dec. 15	37	F. W. Galbraith, Univ. of Arizona, on his prospecting trip to the Yukon.
Carlsbad Potash	Dec. 17	182	Annual Christmas dinner and dance.
Chicago	Jan. 4	75	T. B. Counselman, The Dorr Co., on fluidization of solids in noncatalytic operations.
Cleveland	December	65	L. E. Young, President, AIME; and A. R. Troiano, Case Institute of Technology, on evaluation of transformation characteristics of ferrous alloys.
Colorado	Dec. 14	53	Movie on trackless coal mining.
Columbia	Jan. 6	28	D. W. Smith, Permanente Metals Corp, on Permanente research lab.
East Texas	Dec. 13	43	B. Adamson, Halliburton Oil Well Cementing Co., on a new approach to gun perforating.
Lehigh Valley	Dec. 9	92	Annual meeting; dinner dance.
Montana	Dec. 14	86	Colin Christie on silverware and its effect on civilization.
Tri-State	Dec. 28		Business meeting of newly elected officers.
Utah	Dec. 15	65	M. P. Romney, USSR&M, on what's happened to the prospector.
Washington, D.C.	Dec. 6		L. E. Young, President, AIME.

Fred S. Mulock will succeed **Michael H. Kuryla** as president of the U.S. Smelting Refining and Mining Co. upon Mr. Kuryla's retirement on March 1.

Merritt A. Peterson worked for U.S. Smelting Refining and Mining after graduating from the University of Alaska. Since July he has been employed as assayer-engineer by the Alaska Pacific Consolidated Mining Co., Independence mine, at Wasilla, Alaska.

Roger V. Pierce, general manager of the Cate Equipment Co. in Salt Lake City for the past three years, and from 1936 to 1946 special representative of Ingersoll-Rand's mine hoist and slusher division, has taken a leave of absence from Cate Equipment to handle a mining consulting assignment.

Robert W. Pullen, senior in mining engineering at the Montana School of Mines, has won the $500 award offered by the American Smelting and Refining Co. for his high scholastical standing. His index of 2.7603 is close to the highest possible 3.00.

Gilman C. Ritter graduated from the University of North Dakota in June of 1949 and is employed by the U.S. Smelting Refining and Mining Co. in the engineering section at the Lark mine. His address is 6895 S. 7th E., Midvale, Utah.

J. J. Rutledge is chief consultant to the Maryland Bureau of Mines. He had served for 26 years as chief mine engineer and director of the bureau.

T. Calvert Slater, who has been general superintendent of the Central Eureka Mining Co., is now geologist with the Calaveras Cement Co., Sutter Creek, Calif.

Keith C. Stansmore, assistant manager of the foreign division of The Dorr Co., returned home on Dec. 17 from a three month trip during which he visited many of the mining districts throughout Australia, and the firm of Hobart Duff Pty. Ltd., Melborne, the Dorr Company's representative in Australia and New Zealand.

Robert E. Sykes is working as a mining engineer for Minas de Matahambre, Matahambre, Provincia de Pinar del Rio, Cuba.

Glenn L. Thompson recently resigned as general superintendent of the Alder Gold-Copper Co. of Twisp, Wash., where a 250-ton selective flotation mill was nearing completion. His present address is 205 N. Sawyer St., Olympia, Wash.

William E. Strang has resigned his various positions on the AIME staff to join the production division of the API in Dallas.

H. Newell Appleton became office manager at AIME headquarters on Jan. 1. More detail on his new, and numerous duties, and on Bill's resignation appears in the Drift of Things.

John M. Tomaszewski was graduated from the University of Pittsburgh with a degree in mining engineering last September and is now employed by the Berwind-White Coal Co., Windber, Pa.

Pierce C. Walker, who was manager for Lincoln Mines, Inc., Hiko, Nev., is at Cripple Creek, Colo., working on a new 200-ton mill to treat the gold-silver ores of the Cripple Creek district.

J. W. Woomer, of J. W. Woomer & Associates, consulting mining engineers of Pittsburgh and Wheeling, returned in December from a four-week trip to Australia. His work there covered an inspection of the steel company mines in Eastern Australia followed by an examination of the virgin coal available in Western Australia for the construction of a steel plant in that state. The study involved entering Australia on the east from San Francisco and, after crossing Australia, returning to New York by way of India, Egypt, Rome, and London. The entire trip was made by air.

H. S. Worcester has resigned as president of Telluride Mines, Telluride, Colo. He is succeeded by **E. S. McCurdy,** of San Francisco.

Paul Zinner, chief of the Minneapolis branch of the Bureau of Mines, has been appointed regional director, covering seven states.

Obituaries—

Ralph McLane Bowman (Member 1940), buyer of raw materials for the Republic Steel Corp., died June 18, 1949. During the decade following graduation from Cornell, Mr. Bowman designed bridges for various companies and in 1920 became assistant purchasing agent for the Otis Steel Co., buying mill machinery, supplies, and refractories. In 1925 he joined Republic Steel, buying all refractories for the Corporation, all ferroalloys, other raw materials, making investigations of limestone properties, and methods of manufacture of numerous commodities used in steel plants and blast furnaces.

Aubrey Hamilton Garner (Member 1917), for many years a consulting engineer and geologist, died Nov. 26, 1949, in Dallas. He had been with DeGolyer & MacNaughton and worked quite a bit in Rio de Janeiro. For years he was a partner in the firm, Brokaw, Dixon, Garner & McKee.

William E. Brewster (Member 1933) died suddenly on Dec. 19, 1949, at his home in Chicago. He was born at Iron Mountain, Mich., on June 14, 1889, and after preparing for college at Hotchkiss, entered the Sheffield Scientific School from which he was graduated in 1910.

In September of that year he became associated with the Wisconsin Steel Works of the International Harvester Co. at Chicago, and remained with the company throughout his entire industrial career becoming manager of operations for the steel division, holding this post until he retired in 1948.

Endowed with a keen mind and sound business judgment, he was an effective leader and counselor of men. He considered the developing of young men in his organization just as important and worthwhile as the making of steel. He left his mark in his community, in his company, in the steel industry, in character building institutions, and in his church. Would that every man emulate his example, that of leaving the world just a little better than the way he found it. No finer tribute can be paid William E. Brewster than the fact that he not only earned the respect of all who knew him but he also endeared himself to all who knew him.—*R. A. Lindgren.*

Everend Lester Bruce (Member 1937) Miller Memorial professor of geology at Queen's University, Kingston, Ont., died last October. He had been elected vice-president of the Society of Economic Geologists for 1950 shortly before his death. He was born at Toledo, Ont., in 1884, and earned degrees at Queen's, Columbia, and Wisconsin universities. Dr. Bruce had worked for the Geological Survey and the Whitehall Petroleum Corp., before becoming professor of mineralogy at Queen's in 1920.

John Robert Bryden, Jr. (Member 1946), who was a mining engineer with the Hammond Coal Co. and the Kohinoor Coal Co., died on Nov. 9, 1949. He was 57 years old. After graduating from Yale, he went to work as assistant superintendent for the Racket Brook Coal Co. He spent 35 months in the army and in 1919 became superintendent of the Price-Pancoast Coal Co. He had worked also with the West End Coal Co., Monarch Anthracite Coal Co., Scranton Coal Co., and the Centralia Collieries Co.

Kent Perry Campbell (Member 1920), who went to work for the American Smelting and Refining Co. in 1910, died Dec. 7, 1949. After graduating from the Colorado School of Mines, he went to Velardena, Durango, Mexico, as draftsman for AS&R. For two years he was chemist for Cia. Metalurgica Nacional, returning to AS&R in 1914 as chemist. In 1919 he was assistant superintendent. Last year he was transferred from Chihuahua to the company's general engineering department at El Paso.

Warren W. Currens, with the Passaic Valley Water Commission, Little Falls, N. J., died Aug. 6, 1949. Mr. Currens, who was born in Mattoon, Ill., in 1888, graduated from the Colorado School of Mines in 1912 and that same year went to work for the Ray Consolidated Copper Co. He came East in 1920 and became associated with the Montclair Water Co. and later with the Passaic Consolidated Water Co.

Harvey Hardison (Member 1935), who was chief engineer for the Standard Oil Co. of Texas, died Dec. 8, 1949. During the first World War he served in the infantry and after separation from the army returned to the University of California to get his B.S. degree. In 1921 he was a geologist with the Associated Oil Co., then worked for Pacific Oil, Standard Oil, and the California Co.

John Norman "Pope" Houser (Member 1912), vice-president and general manager of the Tennessee Copper Co., and a director of the Tennessee Copper Co. and the Tennessee Corp., died at Knoxville, Tenn., on Nov. 6, 1949.

Mr. Houser was born on Aug. 12, 1882 at Summit Hill, Pa. He entered the mining business at the early age of 17 when he became associated with a New York financial group which managed mining enterprises. Sagely observing that progress, with commensurate rewards, was more likely to be achieved by workers at the mines, he left New York two years later to obtain first-hand operating experience and knowledge in mining and milling. Going first to Colorado and from there to Mexico, he rose rapidly from foreman to general manager. While at Cananea, the first electrostatic separating plant in Mexico was constructed under his direction. His successful activities at Cananea attracted attention and in 1912 he was put in charge of properties of the American Zinc Co. at Carterville, Mo. Mr. Houser remained at Carterville until 1914 when he went to Mascot, Tenn., as manager of the American Zinc Co. of Tennessee. During this period of changing milling methods probably the first use in this country of copper sulphate as a flotation reagent was made under his management. Soon thereafter he become vice-president in charge of mines of the American Zinc, Lead & Smelting Co.

In July, 1919, Mr. Houser became general manager of Tennessee Copper at a time when the company's personnel was somewhat demoralized as the result of World War I. Blister copper and low-strength sulphuric acid were the only products at that time. A carefully planned program of research and operational studies was inaugurated resulting in increased ore reserves and in the recovery of additional products from the ore, i.e. iron in the form of iron sinter and zinc as a zinc concentrate. Today the technical developments in mining, ore beneficiation, smelting and acid manufacture, as practiced by Tennessee Copper, are well known in the industry.

Mr. Houser was sincerely and deeply interested in human relations in industry. Despite his noteworthy technical accomplishments in mining and metallurgy, he probably will be best remembered for his creative organization work at Copperhill. His inherent honesty and frankness merited and won for him a high degree of enthusiastic service and loyalty from his associates, to whom he never failed to give full credit for work well done. He was intuitively a great leader of men.—*E. N. Westlake.*

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rod and ball mills
concentrating tables
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Box B-3, MINING ENGINEERING

Frank E. Lee (Member 1924) died Oct. 19. He had spent his entire professional career with the Nichols Copper Co., starting work with the Company in 1890 at the Capelton mines. In 1893 he went to the operations at Laurel Hill, N. Y., becoming secretary.

Cecil Hubert Lenhart (Member 1944), who resigned last year as general superintendent of the steel plant of the iron and steel division of the Kaiser Co., Fontana, Calif., died on Oct. 18, 1949 at his home in California. Since leaving Kaiser he had been associated with H. A. Brassert & Co. Mr. Lenhart had worked for Carnegie-Illinois as assistant general foreman and assistant superintendent of blast furnace. In 1942 he joined the Kaiser Co.

William John Lynott (Member 1949), instructor in the geology department at the Institute of Geophysical Technology at St. Louis University, died Nov. 16, 1949. He was born in Vancouver, B. C., in 1913 and was graduated from the University of British Columbia. He was studying geology at Princeton when he joined the RCAF in 1942. After getting out of the service in 1945 he became assistant mining engineer for the British Columbia Department of Mines and three years later went to St. Louis.

Edward J. Newbaker (Member 1912), vice-president and general manager of the Berwind-White Coal Mining Co., died June 10, 1949. He had been active in the coal industry since leaving Lehigh University in 1898 and was associated for many years with the Lehigh & Wilkes-Barre Coal Co.

Addresses Wanted

T. W. T. Atherton, (M'92)
S. K. Bradford, (M'02)
F. J. Dorion, (M'96)
N. E. Franklin, (R'28)
R. Krahmann, (M'34)
E. A. Manners, (M'01)
G. McClurg, (R'28)
J. E. McDevitt, (M'26)
W. Moore, (M'05)
J. Morris, (M'98)
A. H. Pogson, (R'28)
J. W. Reno, (M'87)
G. A. Richard, (M'97)
W. E. Richardson, (R'28)
H. L. Rodgers, (R'28)
T. R. Starkey, (M'07)
H. S. Young, (M'03)

Crispin Oglebay (Member 1922), who had resigned recently as president of Oglebay Norton & Co., died on Oct. 23, 1949. He had been active in the affairs of the company for many years and at the time of his death was chairman of the board. His hobbies included tree conservation and race horses. Three days before his death, one of his horses, Lock and Key, won the $10,000 Chevy Chase handicap and broke the track record.

Herbert Paschke (Member 1927), for many years a research chemist and engineer with the Armstrong Cork Co., is dead. Born in New York in 1879, he attended schools in Guatemala, California, and New York. He worked as a levelman in Havana, Cuba, as a salesman in Los Angeles, did laboratory work for the Tide Water Oil Co., and development work in waterproofing underground structures for Sicilian Asphalt Paving Co. He joined Armstrong Cork in 1910.

Walter James Radford (Member 1924), managing director of the Rimu Gold Dredging Co., Grey River Dredging Co. and the Kaneiri Electric Co., with offices at Hokitika, New Zealand, died at Christchurch, N. Z., on Oct. 14, 1949. He was 65. Mr. Radford had gained considerable alluvial mining experience in Alaska, the States, and South America by the time he went to New Zealand in 1929 to manage Rimu Gold Dredging. He was instrumental in the floating of Grey River Dredging and building the dredge.

Necrology

Date Elected	Name	Date of Death
1937	E. L. Bruce	October 1949
1920	K. P. Campbell	Dec. 7, 1949
1935	Harvey Hardison	Dec. 8, 1949
1928	Louis W. Hill	Apr. 28, 1948
1946	George M. Krauss	Nov. 12, 1949
1949	William J. Lynott	Nov. 16, 1949
1912	E. J. Newbaker	June 10, 1949
1922	Crispin Oglebay	Oct. 23, 1949
1896	George Rice	Jan. 4, 1950
1901	Josiah E. Spurr	Jan. 12, 1940

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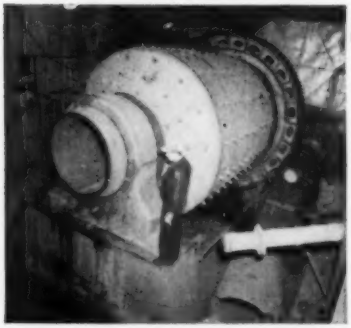

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Joseph C. Yob (Member 1936), manager of the Philipsburg Mining Co., died Feb. 18, 1949 in Santa Monica, Calif. Mr. Yob was born at Hurley, Wisc., 58 years ago. He had worked for Boston Montana Mining and Development Co., the Jib Consolidated Mining Co. and the Jefferson Milling Co. In 1926 he joined the Granite-Bimetallic Consolidated Mining Co., which consolidated with the Philipsburg Mining Co., erecting a 200-ton flotation mill for retreating 100,000 tons of tailings. He also designed and built other flotation mills for the com-

Proposed for Membership

Total AIME membership on Dec. 31, 1949, was 16,315; in addition 4952 Student Associates were enrolled.

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Institute members are urged to review this list as soon as the issue is received and immediately to wire the Secretary's Office, night message collect, if objection is offered to the admission of any applicant. Details of the objection should follow by air mail. The Institute desires to extend its privileges to every person to whom it can be of service but does not desire to admit persons unless they are qualified.

In the following list C/S means change of status; R, reinstatement; M, Member; J, Junior Member; AM, Associate Member; S, Student Associate.

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Ajo—Niemi, Alfred Edwin. (M).

Flagstaff—Riddell, Richard Drake. (C/S—J-M).

Klondyke—Fulmer, J. Walter. (C/S—J-M).

Phoenix—Dutton, Donovan Arnold. (C/S—J-M).

Superior—Sherman, Fred William, Jr. (C/S—J-M).

Tucson—Harshbarger, John William. (C/S—J-M). Williams, Seton Sawyer. (C/S—J-M).

Alabama

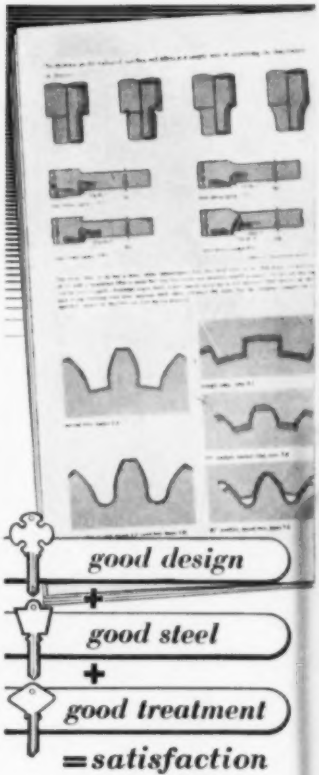
Birmingham—Pagel, Herbert Edwin. (C/S—J-M). Stevens, Edward Henry. (R, C/S—S-M).

California

Bakersfield—Hiatt, William N. (C/S—J-M).

Bellflower—Cox, Eugene Lee. (C/S—S-M).

Glendora—Mull, Bert Hathaway. (C/S—J-M).



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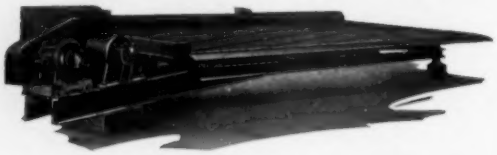

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Westport—Foley, Rupert Merrick. (M).

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Pierce—Wood, Allen Kelsey, Jr. (J).

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Idaho

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Waukegan—Lenz, William Henry. (C/S—J-M).
West Chicago—Kremers, Howard E. (M).

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Gary — Brown, Joseph Andrew. (C/S—S-J).
Logansport — Lawrance, Robert Pryor. (M).

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Chase—Rasmussen, Lyman Merrill. (C/S—S-AM).

Kentucky

Greenville—Lamb, Charles Mitchell, Jr. (J).

Louisiana

Baton Rouge—Manville, Mahlon Francis. (C/S—S-AM).
Homer—Van Horn, Harold Glen. (C/S—S-J).
New Orleans—Vitter, Albert Leopold, Jr. (C/S—J-M).
Shreveport—Morriss, Herbert Alexander, Jr. (C/S—J-M).

Maryland

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Arnold—Ames, John Alfred. (J).

Massachusetts

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New Bedford—Knight, Stanley Storer. (C/S—J-M).
Watertown—Fayerweather, John. (C/S—S-J).

Michigan

Ann Arbor—Ewing, John Frederick. (J).
Negaunee—Boyum, Burton Hill. (C/S—J-M).
Port Huron—Krupp, Roland Gerard. (C/S—J-M).

Minnesota

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Marble—Olson, Reuben Lester. (C/S—J-M).
Minneapolis—Sartell, Jack Albert. (J).
North Hibbing—Wood, Edwin Townsend. (C/S—S-J).

Missouri

Desloge—Riley, Mard Edward. (C/S—J-M).
Hayti — Kidwell, Albert Laws. (C/S—J-M).
Herculaneum—Huddleston, Finis Samuel. (M).
Rolla—Winston, John Stanton. (C/S—S-M).

Montana

Butte—Barrios, Alberto Hoeck. (C/S—S-M).
Great Falls — Eastman, Robert Allan. (C/S—S-J).

Nevada

Boulder City—Jacobs, James Harrison. (C/S—J-M).
Bristol Silver—Werner, Donald G. (AM).
Goldfield — Hargrove, Frances Donald. (M). Hisle, William Hugh. (M).
Round Mountain—Pratt, Morton Edison. (C/S—J-M).
Woodworth—De La Mare, Grover Whitby. (C/S—J-M).

New Jersey

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Morristown — Hearn, Ira Kendell, Jr. (C/S—J-M). Hearn, Jane Kessler. (C/S—J-AM).
Nutley—Hawkins, Jesse Mills. (M).
Perth Amboy—Grodner, Abraham. (M).
Ridgefield Park—Adkins, Robert Charles, (AM).
Upper Montclair — Kuzmick, Jerome Florian. (M).
Westfield—Elliott, John Frank. (C/S—S-J).

New Mexico

Albuquerque — Hillyer, Larry DeWitt. (C/S—S-J). McCroden, Thomas James. (C/S—S-J).
Carlsbad — Arend, Carl A., Jr. (M). Greene, Lloyd Morris. (C/S—J-M). Jameson, Elmer Eugene, Jr. (J). Matheson, Robert K. (C/S—J-M). Murray, Jordan Stephens. (C/S—S-AM). Pressnell, Clarence Eugene. (M). Reeder, Robert Thomas. (C/S—S-J).
Hurley—Winn, William Howard. (C/S—J-M).
Magdalena — Matthews, Gilbert John. (C/S—J-M).

New York

Albany—Platt, Douglas Howard. (M).
Bayside—Maloof, Samuel Barakat. (C/S—S-J).
Brooklyn—Miles, Frederick Victor. (C/S—S-J).
Floral Park—Besnette, Ulys Antoine. (R, C/S—J-M).
Hempstead—Carroll, John Marion. (C/S—S-AM).
Kenmore — Hibbard, David Sharpless. (C/S—J-M).
New York—Cudroff, Louis. (M). Dennis, Richard C. (R—M). Link, Walter Karl. (M). Rynalski, Augustyn T. (M). Sisto, Joseph A. (M). Skinner, Charles F. (C/S—J-M). Willner, Abner Robert. (C/S—S-AM).
Scarborough—Emison, John Clinton. (M).

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TRADEMARK REG.
COPPER-MOLYBDENUM-ALLOY
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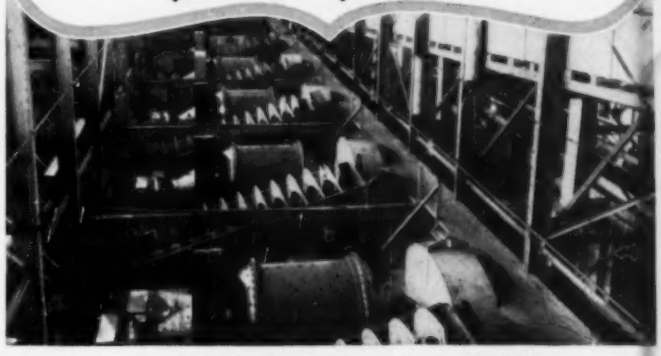
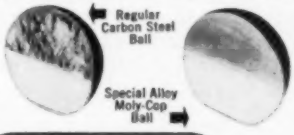


Quincy Mining Company's six 8 ft. x 6 ft. Hardinge Ball Mills on Torch Lake, Hancock, Mich.

Based on a 9 month test period ending June 30, 1949, the consumption of Sheffield Moly-Cop Balls was less than ⅓ as much as forged steel balls previously used by Quincy Mining Company in the six mills shown above. The forged steel balls were consumed at the rate of 1.55 pounds per ton of ore ground as against .49 lbs. of Sheffield Moly-Cop balls.

Actual cash savings were more than $30,000 in nine months, despite the original higher per-ton cost of Moly-Cop balls. The economy of Moly-Cop Balls has been proved in mining operations all over the world.

The harder martensitic structure of Moly-Cop Grinding Balls, their toughness right to the core, are the reasons for this longer grinding life and grinding economy. Your own ore-reduction costs can be reduced considerably, when you charge your mills with Moly-Cop Grinding Balls.



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Chesterland—Mumma, Harries A., Jr. (C/S—J-M).
Cleveland—Irwin, J. Preston. (M).
Columbus—Jaffee, Robert I. (C/S—J-M). Pincus, Howard J. (C/S—S-J).
Euclid—Dennis, Howard J. (AM).

Oklahoma

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Oklahoma City—Eyler, Albert Taylor, Jr. (R, C/S—S-M). Whiteley, Burton W. (C/S—J-M).
Paula Valley—Hardebeck, Edward James. (C/S—S-J).
Tulsa—Chapman, Robert Bruce. (C/S—J-M). Culp, John Douglas. (C/S—J-M). Keller, Wallace O. (C/S—J-M). McMillen, Robert Edward. (C/S—J-M). Moore, John Isler, Jr. (C/S—J-M). Nedom, H. Arthur. (C/S—S-J).

Oregon

Albany—Rasmussen, Robert Thorvald C. (C/S—J-M).
Portland—Dole, Hollis Mathews. (C/S—J-M).
Troutdale—Amos, Lloyd Arthur. (M).

Pennsylvania

Beaver—Peterson, Karl Freeman. (C/S—J-M).
Bethlehem—Good, Robert Charles, Jr. (C/S—J-M).
Duquesne—Wagaer, George Edwin. (M).
Ebensburg—Jones, Sheldon. (C/S—J-M).
Glenshaw—Trout, Harry E., Jr. (C/S—J-M).
Harrisburg—Gray, Carlyle. (J).
Holsopple — Tomaszewski, John Martin. (C/S—S-J).
Lebanon—Dominguez, Ezequiel Chavez. (C/S—J-M).
Mollenauer—Popp, Victor T., Jr. (M).
Nemacolin—Core, Jesse Franklin. (C/S—J-M).
New Kensington — McClintock, Thomas Irvin. (C/S—J-M).
Norristown — Ellington, Thomas Settle. (M).
Palmerton—Held, Llewellyn John. (C/S—J-M).
Pittsburgh—Conroy, Edward Hubert, Jr. (AM). Haines, Harry William, Jr. (J). Lustman, Benjamin. (C/S—J-M). Peterson, John Louis. (R, C/S—S-M). Simnad, Massoud T. (J). Vonfeld, Jacques Martin. (C/S—J-M). Walmer, C(harles) Richard. (AM).
Pleasant Hills—Ogan, Albert C. (M).
Punxsutawney—Bullers, William Irvin. (C/S—J-M).
Scottdale—Anger, Earl Maxwell. (AM).
Secane — Allshouse, Mervin Sorber, Jr. (M).
Temple—Eberly, Warren Samuel. (M).
Uniontown—Opermann, Albert J. (M).

South Carolina

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Tennessee

Copperhill—Flournoy, Ezell. (M).

Texas

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Bellaire—Dumbauld, George Karl. (M).
Chapman Ranch—Davis, William Smith. (C/S—S-J).
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Falls City—Upchurch, Mack. (C/S—J-AM).
Houston — Blair, Philip Michael. (J). Bodin, Jean Alfred. (M). Evans, John Billy. (C/S—J-M). Laas, Charles William. (M). Nichols, James Bryant. (M). Rachford, Henry Herbert, Jr. (J).
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Nitro—McIntosh, Jack Lorne. (C/S—J-M).

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Monterrey, Nuevo Leon—Villareal, Eduardo Mauricio, Jr. (C/S—J-M).

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Cuba

Pinar del Rio—Sykes, Robert Edward. (C/S—S-J).

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Itabira, Minas Geraes—Franca, Joao Mendes. (C/S—J-M).

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Yauricocha—Allen, Kenneth Laidlaw. (J).

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Aldwick Bay—Goulden, Charles Edward. (C/S—S-J).
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Baguio—Dana, John Francis. (C/S—J-M). Farretta, Alvin Joseph. (C/S—J-AM).
Rizal—Boquer, Antonio Maria. (C/S—J-AM).

Advertisers' Index

NEW PUBLISHING SCHEDULE 169

Aero Service Corp. ..... *
The Aitkin-Kynett Co.

Allen-Sherman-Hoff Co. ..... Third Cover
Eugene A. Holland

Allis-Chalmers Mfg. Co. 154, 168
Compton Advertising, Inc.

American Cyanamid Co. ..... *
James J. McMahon, Inc.

American Manganese Steel Div., American Brake Shoe Co. ..... *
Fuller & Smith & Ross, Inc.

Anaconda Copper Mining Co. 164
Kenyon & Eckhardt, Inc.

Christensen Diamond Products Co. ..... *
Axelsen Adv. Agency

Clarkson Mfg. Co. ..... *
Eggers-Rankin Adv. Service

Climax Molybdenum Co. 307
Rowlinson-Broughton

Colorado Fuel & Iron Corp., Wickwire Spencer Steel Div. ..... *
Fuller & Smith & Ross, Inc.

Colorado Iron Works Co. ..... *
Shaw-Schump Adv.

Deister Concentrator Co. 308
Louis B. Wade, Inc.

Denver Equipment Co. 157

Dorr Co., The 174

Dow Chemical Co., The Great Western Div. 156
MacManus, John & Adams, Inc.

Eimco Corp., The ..... *

El-Tronics, Inc. ..... *
Packard Advertising

Euclid Road Machinery Co. ..... *
The Bayless-Kerr Co.

Flexible Steel Lacing Co. ..... *
Kreicker & Meloan, Inc.

Gardner-Denver Co. 161
The Buchen Co.

General Electric Co. 166, 167
G. M. Basford Co.

Goodman Mfg. Co. 155
L. H. Gross and Assoc.

Hardinge Co., Inc. 307
The W. H. Long Co.

Harnischfeger Corp. ..... *
The Buchen Co.

Hendrick Mfg. Co. ..... *
G. M. Basford Co.

Hercules Powder Co., Inc. ..... *
Fuller & Smith & Ross, Inc.

Hercules Powder Co. (Flotation) ..... *
Fuller & Smith & Ross, Inc.

Humphreys Investment Co. ..... *
Ed M. Hunter & Co.

Ingersoll-Rand Co. ..... *
Richard & Co.

International Nickel Co., Inc. 172
Marschalk & Pratt Co.

Jones & Laughlin Steel Corp. ..... *
Ketchum, MacLeod & Grove, Inc.

Joy Mfg. Co. 162, 163
Walker & Downing

Link-Belt Co. ..... *
Don Proctor Adv.

Longyear Co., E. J. 160

Merrill Co. 305
The McCarty Co.

Mine & Smelter Supply Co. ..... *
Shaw-Schump Adv.

Mine Safety Appliances Co. ..... Fourth Cover
Walker & Downing

Morse Bros. Machinery Co. 160

National Technical Laboratories ..... *
Dozier-Graham-Eastman

Nordberg Mfg. Co. ..... *
Russell T. Gray, Inc.

Oliver United Filters Inc. ..... *
Sanger-Funnell, Inc.

Radiac Co., The ..... *
Admiral Adv.

Sauerman Bros., Inc. ..... *
Symonds, MacKenzie & Co.

Sheffield Steel Corp. 309
R. J. Potts-Calkins & Holden Adv.

Texas Gulf Sulphur Co. 159
Sanger-Funnell, Inc.

Traylor Engineering & Mfg. Co. 158
Kamp and Godfrey, Inc.

Tyler Co., W. S. 169

Wellman Engineering Co. ..... *
The Griswold-Eshelman Co.

Western Machinery Co. 171
Walther-Boland Assoc.

Wheel Truing Tool Co. ..... *
Clark & Rickerd, Inc.

Wilfley & Sons, Inc., A. R. ..... Second Cover
Ed M. Hunter & Co.

Wilmot Engineering Co. ..... *
Wilbur A. Myers

Yuba Mfg. Co. ..... *
George C. McNutt Adv.

*Previous Issues

This view of the plant far below gives an idea of the head and long distance involved, and the piping runs still further than shown here.

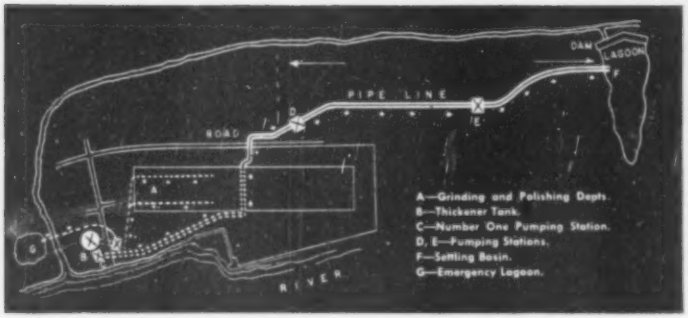

Diagram at left shows entire pumping system. Waste comes from A to thickener, B, then to three pumping stations, C, D, E, and settling basin, F. There are actually two systems, one the duplicate of the other. Hydroseals are in series, two in each station for each system. All pumps are operated by remote control with automatic start and stop. Since piping is cross connected, discharge can be automatically switched from one line to the other or just part of the other. Above photo shows one of the Hydroseals in the first pumping station.

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